Economics of Large-Scale Thorium Oxide Production: Assessment of Domestic Resources

J. K. Young   M. Y. Ballinger
C. H. Bloomster   M. K. Drost
W. I. Enderlin   S. A. Weakley
M. H. Morgenstern

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JK Young
CH Bloomster
WI Enderlin
MH Morgenstern
MY Ballinger
MK Drost
SA Weakley

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1.0 SUMMARY

Current interest in nonproliferation has prompted reassessment of the advantages of the thorium fuel cycle. However, a successful thorium program depends on resource availability, economic viability, and the environmental safety of not only the power plant, but also the associated mining, milling, and refining facilities. To determine the feasibility of the thorium fuel cycle, the supply curve for thorium oxide was developed for domestic thorium resources (see Figures 1.1A and 1.1B). The curve illustrates the relationship between production costs and quantity of thorium oxide available. All production costs are in 1978 dollars. Production cost figures include costs for baseline environmental control.

In the supply curve, the quantity of reserves and the most likely production cost is illustrated by a rectangular area on the supply curve for each deposit. A double arrow drawn perpendicular to the top of the rectangle

SUPPLY OF THORIUM OXIDE, SHORT TONS OF THO$_2$
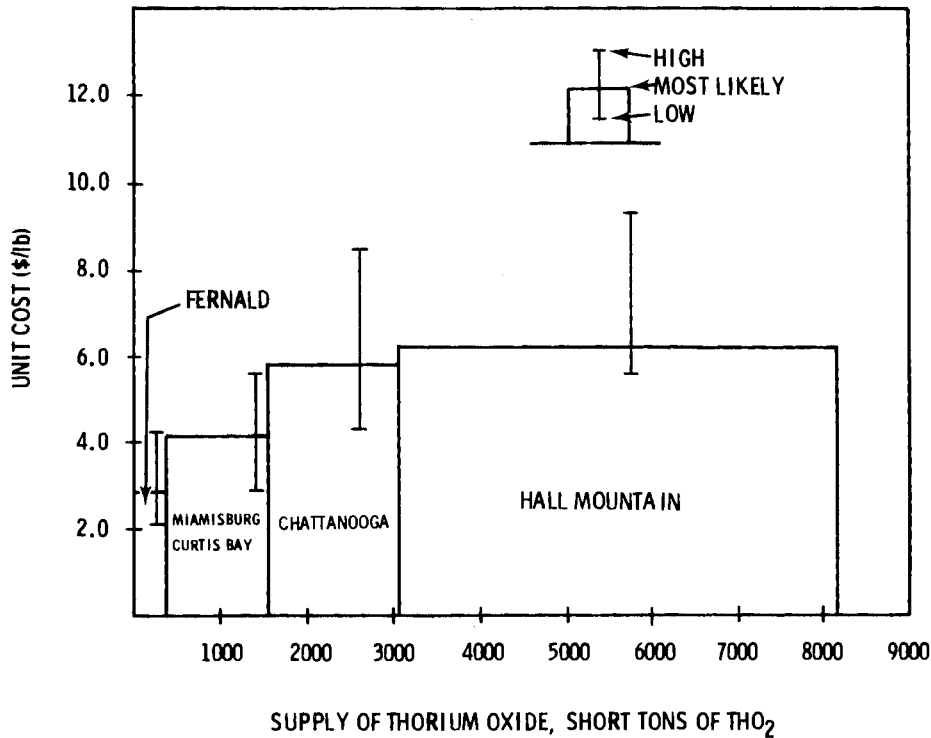
FIGURE 1.1A. Domestic Thorium Supply Curve

1.1
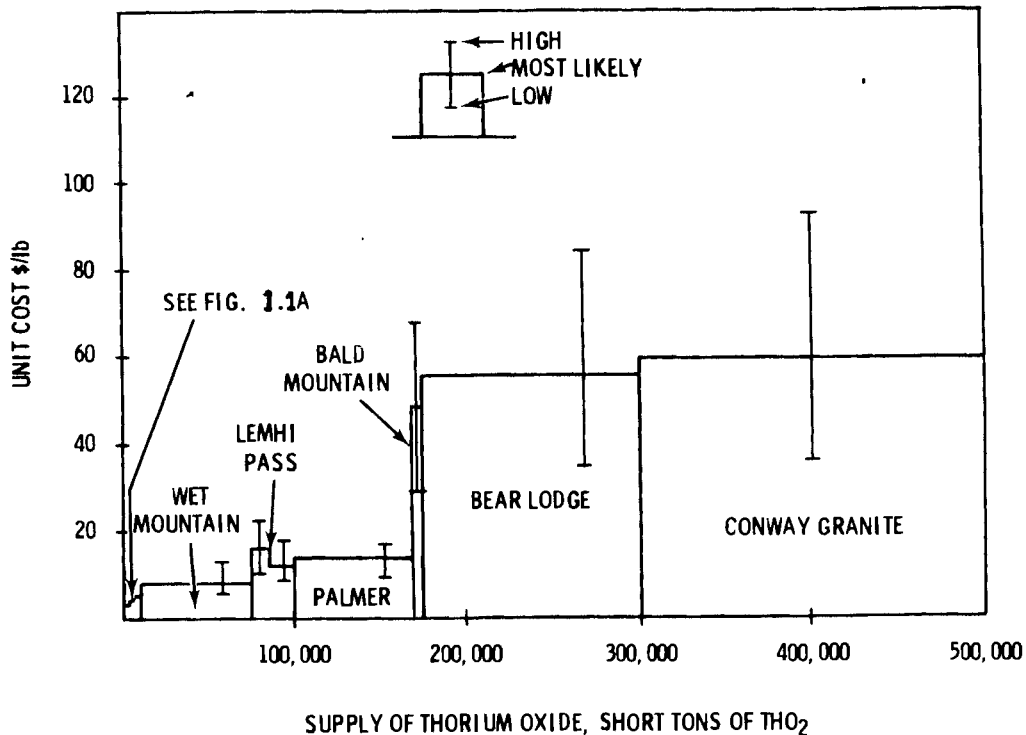
FIGURE 1.1B. Domestic Thorium Supply Curve

illustrates high and low cost estimates for each deposit. Deposits are arrayed by plotting production costs in increasing order except for Lemhi Pass. The Lemhi Pass deposit shows a decrease in cost because Lemhi Pass is mined in two stages. Costs for the first stage of mining are higher because a lower grade of ore is developed. As mining progresses, higher grades of ore are accessed (second stage) and production costs decrease.

The supply curve illustrates that sufficient amounts of thorium exist to supply a domestic thorium-reactor economy. Most likely costs of production range from $3 to $60/lb ThO_2. Near-term thorium oxide resources include the stockpiles in Ohio, Maryland, and Tennessee and the thorite deposits at Hall Mountain, Idaho. Costs are under $10/lb thorium oxide. Longer term economic deposits include Wet Mountain, Colorado; Lemhi Pass, Idaho; and Palmer, Michigan. Most likely costs are under $20/lb thorium oxide. Long-term deposits include Bald Mountain, Wyoming; Bear Lodge, Wyoming; and Conway, New Hampshire. Costs approximately equal or exceed $50/lb thorium oxide.

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The supply curve was developed by characterizing the principal identified domestic thorium resources by size, location, and type and grade of ore. Conceptual production processes were determined for the mining, milling and refining of reactor grade thorium oxide and costs of producing thorium oxide were estimated assuming current environmental control practices.

Thorium recovery processes are similar to uranium. After recovery by underground mining, vein deposits undergo crushing and grinding followed by acid dissolution, solvent extraction, precipitation, and calcination to reactor-grade thorium oxide. Disseminated and granitic deposits undergo chemical processing techniques similar to vein deposits but are mined by open-pit methods and preconcentrated by gravity and magnetic separation. The current source of thorium oxide, stockpiled throium sludge, is in a semi-processed state. It is refined by solvent extraction at Tennessee Nuclear Facilities, Jonesboro, Tennessee or National Lead Company of Ohio, Fernald, Ohio.

Production costs were determined from the capital and operating costs for each processing facility. The fixed charge rate method was utilized to annualize capital costs and to include state and local taxes, depreciation, interest, return on investment, and insurance in the production cost. Depletion and byproduct credit were also determined. Operating costs are comprised of direct and indirect expenses. Direct expenses include labor, maintenance and repairs, utilities, reagents, and supplies. Indirect expenses include payroll overhead, plant overhead, and administrative charges.

Uncertainty in cost estimates is determined using Monte Carlo simulation techniques. From estimates of high, most likely, and low values for plant capital and operating cost, product recovery, byproduct credit, plant size, and plant life, the range of production costs can be determined for each deposit. Costs for thorium including uncertainty range from $2.00/lb to $88.00/lb of ThO_2.

In addition to developing the supply curve, this report identifies effluent releases from the development of various thorium resources. The environmental impact of thorium and costs of various levels of control will be evaluated in a subsequent report.

2.0 INTRODUCTION

The development of nuclear energy is subject to resource availability, economic viability, and political policy. A successful nuclear program depends on an acceptable integration of these factors. To aid in evaluating alternative fuel cycles, data is necessary on the price and sources of nuclear material with time. The effect of policy alternatives on the economic viability of the proposed nuclear fuel cycle can then be evaluated.

In recent years, political opinion has prompted the reassessment of the thorium fuel cycle as an alternative source of power. In a nuclear reactor, thorium breeds the fertile isotope, U-233, rather than plutonium, which is bred in uranium fueled reactors. Either plutonium or uranium-233, depending on the fuel cycle, is separated from the irradiated fuel and mixed with natural or depleted uranium (U-238) to form new fuel. Conceivably, this nuclear fuel material could be diverted by terrorists or foreign governments and utilized for weapons manufacture. Unlike plutonium, however, the fissile isotope U-233 can only be separated from the depleted uranium by an expensive isotopic separations process.

The purpose of this report is to determine the possible range of costs to produce feedstock for fabrication of thorium-bearing fuels. This report derives the long-term supply curve for mining, milling, and refining reactor-grade thorium oxide. The supply curve is developed by first determining a maximum potential demand for thorium. Domestic thorium reserve estimates sufficient to meet this demand are identified based on available data from the U.S. Geological Survey and the U.S. Bureau of Mines plus reserve estimates generated during the course of the study by Pacific Northwest Laboratories operated for the Department of Energy by Battelle Memorial Institute (PNL). The thorium resources are characterized by size, location, type and grade of ore, etc. For each of the identified thorium resources, processes for mining through refining are determined and corresponding production costs are generated. The supply curve is obtained by plotting thorium oxide supply versus the cost of production. Energy costs can be calculated from the cost of the

feedstock, the other costs of the fuel cycle, and the fuel exposures to determine the economic viability of the thorium fuel cycle.

In addition to developing the supply curve, this report identifies effluent releases from the development of various thorium resources. The environmental impact of thorium and costs of various levels of control will be assessed in a subsequent report.

3.0 MAXIMUM POTENTIAL DEMAND

In developing the thorium supply curve, sufficient domestic resources are identified and characterized to meet future thorium oxide demands. The maximum thorium oxide demand, as shown below, is approximately 700,000 tons of thorium by the year 2070. This demand is an upper boundary; it is not a projection.

Maximum potential demand was calculated from growth estimates of total domestic nuclear generating capacity. The nuclear generating capacity estimates were derived by a computer code, ENFORM[1].

Thorium demand was determined by assuming a ten percent conversion of nuclear generating capacity to the thorium fuel cycle annually. Based on our assumption, all nuclear reactors would be converted to thorium by the year 2000, as shown on Table 3.1. Annual thorium demand is calculated by assuming 27 tons per year of thorium oxide are required per 1000 MWe of power. This thorium requirement assumes no recycle of irradiated thorium. From the annual demand, cumulative demand is calculated and maximum potential demand is determined.

TABLE 3.1. Potential Thorium Oxide Demands

Year	Total Capacity (MWe)	Thorium Capacity (MWe)	Annual Thorium Demand (Tons/Yr)	Cumulative Thorium Demand (Tons)
1983	88,160	8,816	238	238
1985	120,365	31,245	843	1,600
1990	184,080	110,385	2,980	11,889
1995	278,065	229,779	6,204	35,948
2000	380,000	380,000	10,260	79,627
2010	480,000	480,000	12,960	199,859
2020	447,149	447,149	12,073	325,237
2030	324,116	324,116	8,751	427,807
2040	248,000	248,000	6,696	502,804
2050	232,001	232,001	6,264	572,785
2060	213,203	213,283	5,758	631,501
2070	208,732	208,732	5,636	694,712

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4.0 PURITY OF FINAL THORIUM PRODUCT

Since this report determines technology and costs of producing nuclear grade thorium oxide, we needed a set of nuclear specifications which satisfy the purity requirements for thorium used in thermal reactors. Currently, no American Standards and Testing Materials (ASTM) specifications exist for reactor-grade thorium oxide, although draft specifications have been prepared. Thus, we developed a set of purity standards which parallel the ASTM specifications for nuclear-grade uranium oxide. Using these standards as guidelines, processes for refining thorium oxide were designed which theoretically produce a satisfactory thorium product.

4.1 DEFINITION OF REACTOR-GRADE THORIUM OXIDE

A hypothetical standard for thorium oxide utilized for nuclear purposes is summarized in the following sections. Only those requirements which affect processing techniques are discussed.

4.1.1 Chemical Requirements

Thorium content, utilizing current processing techniques, is a minimum 99% thorium oxide on a dry weight basis. Thus, a reactor specification of 99% thorium oxide shall be assumed. Assuming ASTM specifications similar to uranium oxide, chemical impurities shall not exceed the individual element specification shown on Table 4.1 and the summation of impurities shall not exceed 1500 ppm.

4.1.2 Nuclear Requirements

In order to obtain a favorable neutron economy, reactor-grade thorium must be free from elements with high thermal neutron absorption cross-sections. For thermal reactor use, the total equivalent boron contributions (EBC) are utilized to determine acceptable levels of parasitic neutron absorbers. Assuming the same specifications exist for uranium oxide powder as for thorium oxide powder, the total EBC must not exceed 4.0 ppm (Table 4.2). As with uranium oxide fuel, isotopic limits shall be specified by the purchaser. This is of special concern when dealing with thorium from ores associated with uranium (Appendix C).

4.1

TABLE 4.1. Chemical Impurities in Nuclear Grade Thorium Oxide

Element	Maximum Concentration Limit ppm[a]	Actual[b] Conc. ppm
Aluminum[c]	100	<5
Carbon	100	N/A
Calcium + Magnesium[c]	200	110+
Chlorine + Flourine[c]	350	20+
Chromium	200	N/A
Cobalt	100	N/A
Copper[d]	250	5
Iron[e]	250	1
Lead	250	<5
Manganese[d]	250	10
Molybdenum	250	N/A
Nickel	200	10
Nitrogen	200	N/A
Phosphorus[e]	250	.12
Silicon[e]	200	60
Tantalum	250	N/A
Tin	250	N/A
Titanium[f]	250	.5
Tungsten	250	N/A
Vanadium	250	N/A
Zinc	250	N/A
Uranium[f]	10	.07

a. Annual Book of ASTM Standards for "Nuclear-Grade Sinterable Uranium Dioxide Powder", Part 45, C753.

b. Actual measured impurity levels for thorium oxide obtained using solvent extraction methods of purification.

c. V. S. Yemel'Yanov and A. I. Yevstyukhin, The Metallurgy of Nuclear Fuel, Pergamon Press Ltd., Oxford, London, P. 401, 1969.

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e. R. J. Callow, The Industrial Chemistry of the Lanthanonons, Yttrium, Thorium and Uranium, Pergamon Press Ltd., Oxford, London, P. 182, 1967.

f. Estimate.

N/A Not available.

TABLE 4.2. Parasitic Neutron Absorbers in Nuclear-Grade Thorium Oxide

Element	Absorption Cross Section (barns-2200 m/s)$^{\sigma}$a	Atomic Weight	EBC Factor	Conc. in ThO$_2$ ppm	EBC[b]
Aluminum[c]	.235	26.98	.0001	<5	.0005
Barium	1.2	137.34	.0001		
Boron[c]	759	10.81	1.0000	.05	.05
Cadmium[b]	2450	112.40	.3104	.63	.196
Cesium	30.	132.91	.0032		
Chlorine[e]	33.2	35.45	.0133	<20	.266
Chromium	3.1	52.00	.0008		
Cobalt	37.2	58.93	.0090		
Copper[d]	3.8	63.54	.0009	5	.0045
Dysprosium[c]	930	162.50	.0815	<.08	.007
Europium[c]	4400	151.96	.4124	.02	.04
Gadolinium[c]	49000	157.25	4.438	.12	.53
Hafnium	105	178.49	.0084		
Iron	2.55	55.85	.0007	1	.0007
Lithium[c]	71.0	6.939	.1457	<.05	.35
Manganese[d]	13.30	54.94	.0034	10	.034
Molybdenum	2.70	95.94	.0004		
Nickel[d]	4.60	58.71	.0011	10	.011
Nitrogen	1.85	14.01	.0019		
Phosphorous[e]	.19	30.97	.0001	60	.006
Samarium[e]	5820	150.35	.5513	<.12	.066
Silicon[e]	.16	28.09	.0001	60	.006
Tantalum	21	180.95	.0017		
Tin	.63	118.69	.0001		
Titanium[f]	6.1	47.90	.0018	.5	.0009
Tungsten	18.5	183.85	.0014		
Vanadium	5.06	50.94	.0014		
Zinc	1.1	65.37	.0002		
				TOTAL	1.569 $\frac{ug}{g}$

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[a] Brookhaven National Laboratory Publication BNL-325, Second Edition, July, 1958 and Supplement No. 1, January, 1960, and Supplement No. 2, 1964-1966.

[b] The EBC is calculated for each element as follows:
   EBC of impurity, ug/g = $(A_{impurity})$ (ug impurity/g thorium)
   where:
   $A_{impurity}$ = (atomic weight of boron)($\sigma_a$impurity)/($\sigma_a$boron)(atomic weight of impurity)
   where:
   σ_a = 2200 m/s thermal neutron absorption cross section, barns.

   For fast reactor use, the above EBC restrictions do not apply.

[c] V. S. Yemel'Yanov and A. I. Yevstyukhin, The Metallurgy of Nuclear Fuel, Pergamon Press Ltd., Oxford, London, P. 401, 1969.

[d] Prakesh, B. et al., Metallurgy of Thorium Production, Developments in Peaceful Applications of Nuclear Energy, No. 22, International Atomic Energy Agency, Vienna, P. 24, 1963.

[e] R. J. Callow, The Industrial Chemistry of the Lanthanonons, Yttrium, Thorium and Uranium, Pergamon Press Ltd., Oxford, London, P. 182, 1967.

[f] Estimate

4.1.3 Physical Requirements

Bulk density and particle-size requirement will probably be established. For purposes of this report, a particle size of less than 75um will be required along with an average particle size of less than 10um. Bulk density shall be a minimum of 1.0 g/cm^3 as determined by the Scott Volumeter technique.

4.2 ACTUAL PRODUCT PURITY

In determining whether a product constitutes reactor-grade ThO_2, major concern is with the removal of elements with high absorption cross-sections. Elements which cause the most problem are Gadolinium, Europium, Samarium, Cadmium, Dysprosium, Lithium, and Boron. This is a concern in LWR fuels.

For most processes in this study, adequate data on product purity is not available. Information on important impurities contained in thorium oxide from monazite sand is published, however. Data in Tables 4.1 and 4.2 indicates that thorium processed similar to methods conceptualized in this study meets conceptual specifications. Since ores other than monazite also utilize similar refining techniques, the end product should meet reactor-grade quality for all ores studied in this report.

5.0 THORIUM ORES AND THEIR PROCESSING TECHNOLOGY

5.1 CURRENT THORIUM INDUSTRY

Currently, demand for thorium is primarily in the gas mantle industry. Other uses include magnesium-based high temperature alloys for aircraft and aerospace applications, in refractories, for chemical, electronic, and metallurgical applications, and in research. In 1977, ten tons of ThO_2 was also used in two nuclear reactors: General Atomic's 330 MWe high temperature gas cooled reactor at Fort St. Vrain, Colorado and the Department of Energy's 50 MWe experimental light water breeder reactor at Shippingport, Pennsylvania. The price for thorium was $2.75/lb of nitrate and $17.50/lb of nuclear grade ThO_2 in 1977[1].

Monazite sands feedstocks are utilized to obtain ThO_2. Australia, India, Brazil, and Malaysia supply about 81%[2] of the annual U.S. monazite demand. Monazite is produced domestically as a byproduct of heavy mineral production by Humphrey's Mining Company located in Florida. The cost for monazite in 1977 was approximately $164/ton[1].

Domestically, monazite is processed primarily for rare earths by Davison Chemical Division of W. R. Grace and Company, Chattanooga, Tennessee. The monazite is treated with caustic soda and the digested solution is neutralized with hydrochloric acid. The thorium-bearing sludge is precipitated and stored in a pond. The rare earths are concentrated and sold in the form of rare earth chlorides. Current thorium processing capacity is approximately 250 tons/year ThO_2 byproduct[2].

Commercial refining facilities include Tennessee Nuclear Specialties which processes thorium sludge for nuclear purposes utilizing solvent extraction with TBP and oxalate precipitation. According to plant management, the capacity of the plant is about 700 lb/day of ThO_2. The plant currently produces 4 batches/yr of ThO_2 at 5000 lb/batch (20,000 lb/yr). The plant reportedly is in operation two 5-day weeks per batch or 8 wks/yr[2].

National Lead Company of Ohio (NLO), a Department of Energy facility, also has the capability of processing thorium sludges and producing

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reactor-grade ThO_2. NLO utilizes solvent extraction with DAAP (diamyl-amyl phosphonate) to purify thorium. The thorium mixture is then precipitated using ammonium hydroxide. Current thorium processing capacity is approximately 1000 kg/day.

5.2 FUTURE THORIUM INDUSTRY

Today domestic thorium production is only about 60 to 100 tons per year,[2] but future nuclear requirements might increase demand drastically. More wide-spread production of thorium-bearing minerals will then be necessary. And a new industrial structure will evolve to process thorium from various domestic resources. The price of thorium oxide will be dictated by the costs of exploiting now undeveloped reserves.

In order to clarify the discussion of future industries, a broad overview of the likely processing methods and costs is presented in this section. A hypothetical structure of the thorium industry is formulated which provides a basis for cost analysis.

5.2.1 Stockpile Processing Stream

Thorium is available in a semi-processed state from both government and industrial stockpiles. Stockpiles include the W. R. Grace pond in Chattanooga, Tennessee and government stocks in Curtis Bay, Maryland; Miamisburg, Ohio; and Fernald, Ohio. (Data on Curtis Bay is dated. Last report indicates the pond was bulldozed over.) A refining process is used to recover thorium from the stockpiles. The conceptual refinery uses solvent extraction to purify the thorium. The thorium is then precipitated as an oxalate and calcined. Reactor-grade ThO_2 is produced. Production costs are calculated to range from approximately \$3.00/lb to \$6.00/lb of ThO_2.

5.2.2 Thorite Processing Stream

Thorite-bearing vein deposits are some of the highest grade domestic sources of thorium. Any large-scale demand for thorium will certainly result in the exploitation of these resources.

Thorium is reported in at least seventeen vein districts in the United States. Larger, better known districts include 1) Lemhi-Pass Idaho-Mont.,

2) Wet Mountains. Colo., 3)Powderhorn, Colo., 4) Hall Mountain, Idaho, 5) Bear Lodge, Wyo. and 6) Mountain Pass, Calif. This study evaluates production costs for the Lemhi Pass and Wet Mountains deposits since they contain eighty-seven percent of total reserves and probable potential resources for vein deposits. The Hall Mountain deposit was also included because of the extremely high tenor of the ore and its importance as a near-term potential source of thorium. Diamond Creek, Bear Lodge, Mountain Pass, and Powderhorn deposits were not evaluated in this study because of the small quantities of reserves available from these districts.

To determine costs, a model underground mine was developed based on the thorite deposits at Lemhi Pass. The mining methods used include resuing and cut and fill stoping depending on vein width. The mine model was also adapted to the Hall Mountain and Wet Mountains deposits.

The conceptual thorite milling process was based on a laboratory model developed by the U.S. Bureau of Mines[3]. Thorite is milled by grinding the ore and leaching with acid. After solvent extraction and ammonium hydroxide precipitation, the precipitate is calcined, producing reactor-grade ThO_2 powder. Mills are located near the mines in order to minimize transportation costs. Production costs range from $6.00/lb to $16.00/lb ThO_2. Costs including uncertainty range from $5.00/lb to $23.00/lb ThO_2.

5.2.3 Monazite Conglomerate Processing Stream

In conglomerates, monazite is dispersed in rock. Thus, the ore must be ground to release the monazite mineral. The two principal deposits are Palmer, Michigan and Bald Mountain, Wyoming. The conglomerate monazite is refined much like monazite from placer sands, but mining and milling are quite different. Production costs are estimated to range from $14.00 to $50.00/lb of ThO_2.

Three facilities are required to process the conglomerates--a mine, a mill, and a refinery. A conceptual open pit mine model was developed to determine costs for the Palmer, Michigan deposit. This model was then adapted to the Bald Mountain deposit. Conceptual milling and refining processes were also determined. The ore is sent to a mill located at the mine site. In the mill

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the ore is crushed and ground. After gravity and magnetic separation, a monazite concentrate is obtained. The concentrate is sent to a nearby refinery where it is mixed with acid to dissolve the thorium. The thorium is removed from the acid solution by solvent extraction. After treatment with caustic soda, the thorium hydroxide is dried, calcined to the oxide, and packaged. The product is reactor-grade ThO_2 powder.

5.2.4 Carbonatite Processing Stream

Identified carbonatite districts include Magnet Cove, Arkansas; Iron Hill, Colorado; Elk Creek, Nebraska; and Mountain Pass, California. Little data exists on the Magnet Cove, Mountain Pass, and Elk Creek deposits. The Iron Hill deposit is too low-grade to be economically mined for thorium alone. However, Iron Hill also contains significant amounts of niobium, uranium, and rare-earth oxides which could be recovered as byproducts. The economic effectiveness of byproduct recovery depends on the extent the saleable materials occur together in the ore. For example, niobium is contained solely in the pyrochlore. Minerals that potentially contain thorium, uranium, and rare earths include pyrochlore, bastneasite, and monazite. Until more data is available on the feasibility of obtaining a thorium concentrate that also contains a high content of byproducts, an accurate cost estimate for a multi-product thorium facility is difficult to determine.

5.2.5 Monazite Sand Processing

Monazite-bearing heavy mineral deposits occur as beach and stream placer sands. The deposits include the Piedmont Placers in North and South Carolina. In these deposits the thorium-bearing monazite sand represents less than one percent of the placer sands; consequently, large amounts of material must be mined to recover a relatively small amount of monazite. The monazite sand usually occurs in conjunction with other valuable heavy mineral sands such as ileminite, rutile, zircon, columbite, and euxenite. In addition, monazite contains rare earths which are potentially salable byproducts. Estimated costs for producing ThO_2 from these placers are more than $100/lb. Thus, these placers are not economical to exploit within the time frame of this study.

Beach Placers in Florida and California are too low-grade to be economically mined for thorium alone. Thorium can only be produced as byproduct. The amount of thorium produced from beach placers will demand entirely on the market for rare earths. Costs for developing thorium from these resources a byproduct is reflected in the cost of developing the Chattanooga, Tennessee sludgepile.

Disseminated deposits include Conway Granite of New Hampshire; Bear Lodge Mountains, Wyoming; and Powderhorn district, Colorado. Resource calculations for these deposits are highly speculative. Because these deposits are important as long-term resources of thorium, the two largest deposits were included in this study. This includes the Conway Granites and the Bear Lodge Mountains.

5.2.6 Disseminated Deposit Processing Stream

Conway granite is a large, but expensive, source of thorium. Conway granite has a relatively high concentration of thorium (50 ppm). The difficulty in mining, crushing, and acid leaching granite results in high capital and operating costs. Conway granite contains low concentrations of uranium (9 ppm) which could be recovered and sold to reduce the cost of the thorium.

While granite has not been mined commercially, the Department of Energy has extensively studied the processes necessary for mining and refining granite. Design and cost estimates for developing granite mining and refining facilities are taken from Department of Energy documents[4,5]. Production costs are estimated to range from $50.00/lb to $60.00/lb ThO_2.

The conceptual granite processing stream consists of an open pit mine, a mill, and a refinery. The mill is located near the mine. The ore is crushed, ground and leached with acid. After a solvent extraction and treatment with caustic soda, a thorium/uranium concentrate is obtained. This concentrate is shipped to a refinery where the uranium is removed and the thorium is further purified by another solvent extraction procedure.

Bear Lodge is a large source of thorium located in Wyoming. Principal identified thorium bearing minerals are brockite, thorite, and monazite. Only

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the thorite mineral would be attacked by the dilute acid leach utilized to recover thorium. Thus the resulting low thorium recovery considerably reduces the economic attractiveness of this deposit.

The deposit is mined similar to the Palmer, Michigan deposit using open pit methods. Thorium ore is crushed, ground and submitted to a dilute sulfuric acid leach. After solvent extraction and ammonium hydroxide precipitation, the precipitate is calcined producing reactor ThO_2. A central mill is located near the three potential open pit mines at Bear Lodge.

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5. F. J. Hurst, et al., Estimated Costs for Recovery of Thorium and Uranium. ORNL-3928, Oak Ridge National Laboratory, Oak Ridge, TN, 28 pp., November 1966.

6.0 PRODUCTION COST MODEL

The objective of the cost model is to derive production costs of ThO_2 for ores in this study. The cost model applies primarily to thorium processing facilities; however, selected elements are also applied to the mining costs determined by a consulting firm. All costs are in 1978 dollars and cost indices for escalation were determined from the U.S. Department of Labor and Statistics.

Production costs for all facilities are determined from estimates of capital and operating costs. The following equation is used:

$$PC = OPC + \Sigma[DCI_i \times FC_i) + (OTH_j \times FC_j)] - DP$$

where:

$\quad PC$ = annual production costs

$\quad OPC$ = annual operating costs

$\quad DCI_i$ = depreciable capital investment of class i where class i refers to all assets with the same depreciable life

$\quad FC_{i,j}$ = fixed charge rate applicable to class i or j

$\quad OTH_j$ = non-depreciable capital investments

$\quad DP$ = depletion allowance

The annual production cost obtained above is divided by the annual production of thorium oxide.

6.1 CAPITAL COST MODEL

Capital investments are required for the mine, mill, and refinery. Major elements of capital cost are shown in Table 6.1. Depreciable capital investments refer to assets which depreciate with use. Nondepreciable assets include land and working capital.

6.1.1 Buildings and Equipment Cost

In this study, buildings and equipment capital costs are calculated from the purchased equipment cost (Table 6.2). Overhead expenses for installation, service facilities, land, various engineering and construction

TABLE 6.1. Components of Capital Investment

Buildings and Equipment Cost
    Main Plant
    Effluent Control
    Auxiliary Plant
Contingency
Tailings Pond[a]
Environmental Impact
Exploration[a]
Feasibility Study[a]
Development
Spare Parts Inventory
Access Road[a]
    Total Depreciable Capital Investment
Total Land Cost
Working Capital
    TOTAL CAPITAL INVESTMENT

TABLE 6.2. Estimation of Buildings and Equipment Cost
(Showing Individual Components)

Component	Basis
Purchased Equipment	Design Estimate
Field Erection	14% of Purchased Equipment Cost
Foundations	12% of Purchased Equipment Cost
Piping - Chuting	22% of Purchased Equipment Cost
Electrical	19.5% of Purchased Equipment Cost
Instrumentation	5% of Purchased Equipment Cost
Buildings	35% of Purchased Equipment Cost
Site Preparation	5% of Purchased Equipment Cost
Service Facilities	55% of Purchased Equipment Cost
Land	6% of Purchased Equipment Cost
Physical Plant Cost	
Engineering and Construction	25% of Physical Plant Cost
Direct Plant Cost	
Contractor's Fee	5% of Direct Plant Cost
Total Plant Cost	
Land (Subtract from Plant Cost)	
TOTAL BUILDINGS AND EQUIPMENT COST	

costs, and the contractor's fee are determined as a percentage of the costs of the processing equipment. As shown on Table 6.2, nondepreciable land costs are subtracted from the total plant costs to obtain a buildings and equipment cost. The buildings and equipment cost is derived for the: (1) main processing plant, (2) effluent control facilities, and (3) auxiliary plants. The buildings and equipment cost is included in the capital cost shown on Table 6.1 and derived in the following sections.

6.1.1.1 Purchased Equipment Cost

One of the major capital costs is equipment. The amount and size of equipment was determined from conceptual flow diagrams, material and energy balances, and reasonable design assumptions. Process equipment cost was obtained from vendor estimates, costs of similar equipment, and industrial estimating guides[1].

6.1.1.2 Equipment Installation

Equipment installation includes costs for field erection, foundations, piping and chuting, electrical work and instrumentation. Field erection includes costs for transportation to the site and installation labor, and is estimated at 14% of the purchase cost. The cost for foundations and supports is estimated at 12% of the purchased equipment cost and includes the cost for concrete and other materials.

The cost for piping and chuting covers valves, fittings, pipe, supports, chutes and labor. Piping for raw materials, intermediate products, steam, water, air, sewer, and others is included in the estimate. Process plant piping is assumed to be 22% of the purchased equipment cost.

Electrical installation consists of: (1) power wiring, (2) lighting, (3) transformation and service, and (4) instrument and control wiring. Costs result from labor, and materials and are estimated at 19.5% of the purchase equipment cost.

Instrumentation includes instrument costs, expenses for auxiliary equipment, installation labor, and materials. For this study, instrumentation is estimated at 5% of the purchase equipment cost .

6.3

6.1.1.3 Buildings

The cost for buildings includes costs for labor, material, supplies, and costs for installation of plumbing, heating, lighting, and ventilation. It is estimated at 35% of the purchase equipment cost.

6.1.1.4 Site Preparation

At 5% of the purchased equipment cost, this includes costs for yard improvements such as fencing, grading, roads, sidewalks, railroad sidings and landscaping.

6.1.1.5 Service Facilities

Service facilities cost includes utilities for supplying steam, water, power, compressed air, and fuel. Also included are facilities for waste disposal, fire protection, first aid, and cafeteria equipment. This is approximately 55% of the purchased equipment cost.

6.1.1.6 Land

Land costs include actual acreage along with accompanying surveys and fees. Land is assumed to be 6% of the purchased equipment cost. Since land is not depreciable, it is treated separately from the buildings and equipment cost.

6.1.1.7 Engineering and Construction

This category includes charges for design, field supervision, construction labor, and inspection. A percentage of 25% of the physical plant cost is used in this estimate.

6.1.1.8 Contractor's Fee

For this study the contractor's fee is estimated at 5% of the direct plant cost.

6.1.2 Contingency

To compensate for design errors or changes, unpredictable events, and other unforeseen expenses, this factor is included in the estimate of capital investment and is 15% of the direct plant cost[2].

6.1.3 Tailings Pond

Tailings ponds are comprised of embankments placed on the ground surface that are required to retain slurries of waste and water. With serious radioactive pollutants such as thorium and uranium, a clay core dam is necessary to control seepage and allow water to evaporate.

Tailings pond cost includes factors for land, tailings pumps, site preparation, and engineering construction fees, tailings decommissioning costs, and spray equipment to control leach dust. Reclamation costs have not been included.

6.1.4 Environmental Impact Assessment

The assessment of the environmental situation at the proposed site is an important factor and should also be included in the estimate of capital cost. The cost includes charges for legal counsel, public hearings and environmental studies. The cost is estimated to be in the range of $50,000 to $100,000 for a 200 tpd mill[2]. For simplicity, our study shall assume a cost of $100,000 for environmental assessment unless better data is available from the Lemhi Pass or Palmer, Michigan models (Appendixes D and E).

6.1.5 Feasibility Study

Feasibility studies are conducted before a program begins. Perspective markets for the product are analyzed, laboratory data is determined, and process methods are selected. After capital and operating cost estimates are determined, project feasibility can be evaluated.

Cost for feasibility studies would, of course, vary from site to site. However, for simplicity, feasibility study costs for facilities (except the mines) are estimated at $100,000[3].

6.1.6 Exploration

Exploration costs for mine developments are incurred which determine the extent, amount, location, grade, or quality of the mineral deposit. These costs include expenses for land acquisition, core drilling, assaying, engineering fees, geological fees, exploratory shafts, pits, drifts, etc.

6.5

Exploration costs are very site specific. They are determined for Lemhi Pass, Idaho and Palmer, Michigan deposits. Depending on expected mining methods, the costs are adapted to other deposits in this study.

6.1.7 Development

Development costs include all expenses incurred when making the ore body accessible. This includes expenses for shafts, raises, all mine headings, and depreciation on equipment used for such work. This cost varies from site to site and is determined similarly to exploration costs.

6.1.8 Access Road

The cost for access roads is site dependent. For simplicity, access roads are estimated at $100,000[3].

6.1.9 Total Land Cost

The summation of land costs for the main plant, auxiliary plant, and effluent control equipment is called the total land cost.

6.1.10 Working Capital

Working capital includes money invested in for: (1) raw materials and supplies, (2) inventory, (3) materials-in-process, (4) accounts receivable, (5) cash on hand, and (6) accounts and taxes payable. Working capital is assumed to be approximately 15% of the fixed capital investment[2].

6.2 OPERATING COST MODEL

The unit cost of ThO_2 production is determined from the annual operating costs. Operating costs include operating labor, supervisory and clerical labor, maintenance and repair, utilities, operating supplies, laboratory charges, chemicals and solvents, and plant and administrative overheads. The basis for estimating these components is discussed below.

6.2.1 Operating Labor

Operating labor is estimated from data from a similar industry or from the labor requirements for process equipment operation[1]. Wage rates, based on data from the U.S. Department of Labor[4], are assumed to be $8.06/hr. unless otherwise specified.

6.2.2 Direct Supervisory and Clerical

The cost for direct supervisory and clerical is assumed to be about 15% of the cost for operating labor.

6.2.3 Maintenance and Repairs

These expenses include the cost for labor, materials and supervision. The total plant cost per year for maintenance and repairs shall be assumed equal to 6% of the capital investment[5].

6.2.4 Utilities

The cost of utilities includes the costs for steam, electricity, fuel oil, and water. The electricity, fuel oil, and water are purchased, but steam is produced on site. Steam costs would be included in charges for fuel oil and water.

Electric power was determined from process equipment requirements. A power factor of .90 and a distribution efficiency of .9 were also assumed. Power cost is assumed to be $.03/kWh.

Fuel oil demand for space heating and process equipment is also calculated. A cost of $0.50/gal. for No. 6 fuel oil was assumed.

Water demand is determined from a material balance around the facility. The cost of water is assumed to be $.05/1000 gallons for city water, or $.15/1000 gallons for well water. The type of water depends on the proximity to an urban area.

6.2.5 Operating Supplies

Operating supplies include such items as charts, lubricants, janitor supplies, test chemicals, etc. The annual cost for operating supplies is about 15% of cost for maintenance and repairs .

6.2.6 Laboratory Charges

The cost of laboratory tests and labor is taken as 15% of the operating labor cost[5]. This cost is necessary to insure proper control of product quality.

6.2.7 Chemicals and Solvents

Chemicals and solvent requirements are determined from material balances for each specific facility obtained from published data on chemical prices.

6.2.8 Plant Overhead

Plant overhead includes costs for plant upkeep and overhead, payroll overhead, packaging, medical services, safety and protection, restaurants, recreation, salvage, laboratories, and storage facilities. Plant overhead is assumed to be 60% of cost for operating labor, supervision, and maintenance.

6.2.9 Administrative Overhead

Administrative costs include costs for executive salaries, clerical wages, legal fees, office supplies, and communications. It is assumed to be 15% of costs for operating labor, supervision, and maintenance[5].

6.3 FIXED CHARGE RATES

A fixed charge rate method is used to simplify the discounted cash flow calculations (see Appendix B). It converts depreciable capital investment into an equivalent annual revenue requirement. Rather than calculate a year by year cash flow, a single fixed charge rate is assumed to apply over the life of the facility. The fixed charge rate includes provisions for capital recovery, bond interest, return on equity, federal income tax, state income tax, property tax and property insurance.

The fixed charge rate was calculated for the facilities assuming they are owned and operated by a mining or chemical processing company. For such a company, the financial and tax data in Table 6.3 was assumed. The fixed charge rate was calculated according to the procedure described in Appendix B. Fixed charge rate varies with plant lifetimes. For non-depreciable assets such as land and working capital, a fixed charge rate of 10% was assumed.

TABLE 6.3. Financial and Tax Assumptions

(%)

Capitalization	
Debt	24
Equity	76
Bond Interest Rate	8
Return on Equity	15
Federal Income Tax	48
State Income Tax	7
Property Tax	2.5 of capital cost
Property Insurance	.5 of capital cost
Mine-Mill Life	Site Dependent
Depreciation	Sum of the years digits

6.4 DEPLETION ALLOWANCE

Depletion is a periodic non-cash expense charged to income to reflect the decline in the value of property. In this study, the percentage method is utilized to calculate depletion. Thorium production qualifies for a depletion allowance of 22% of the gross revenues. According to Internal Revenue Code 613(b), the depletion allowance cannot exceed 50% of income. For the purposes of this study, we assumed that this income test would be met and the full depletion allowance would apply.

The depletion allowance is applicable to the gross revenue which comes from the extraction of thorium ores and minerals in the ground. Only revenues from the mining and concentrating processes would probably qualify for the depletion allowances. The depletion allowance is treated as an expense for income tax purposes which reduces the tax liability.

6.5 BYPRODUCT CREDIT

For many of the ores in this study, byproducts are produced during some stage of processing. In order to determine byproduct credit, the amount of byproducts, the market price of byproducts, and the incremental cost of byproduct production must be specified. The amount of byproducts is determined by

6.9

the composition of the ore and the efficiency of recovery processes. The current market price of byproducts is available in literature and is utilized when potential byproduct production is small relative to conventional supplies. When byproduct production will saturate the market, a credit of less than current market value is assumed. (See Appendix F.) The incremental cost of byproduct production is determined from associated capital and operating costs.

The reduction in thorium production costs resulting from a byproduct credit is determined by Equation 6.1. Byproduct revenues less the incremental byproduct production costs are divided by the thorium production to determine the byproduct credit. If the incremental cost of recovering byproduct is greater than the revenue obtained, no byproduct credit will be considered.

$$BC = [(AMBY \times MP) - INC]/TP \qquad (6.1)$$

where:

BC = Byproduct credit

AMBY = Amount of byproducts

MP = Price of byproducts

INC = Incremental cost of byproduct production

TP = Thorium production

6.6 ROYALTIES

Since royalties affect market price rather than production costs, no provisions for royalties shall be included in this study.

6.7 TRANSPORTATION

According to Title 49 in the Code of Federal Regulations, natural thorium constitutes a low specific activity radioactive material and must be transported in exclusive use vehicles and packaged in strong, tight packages that allow no leakage. Costs for this study were determined assuming railroads are utilized for long distance shipping. Trucks ship thorium to nearby facilities and to and from the rail car.

6.8 SCALING FACTOR

A scaling factor is required to adjust one set of estimates to a different design size. Use of a scaling factor is illustrated by the following equation:

$$C_n = r^s C$$

where:

C_n = new plant cost
r = ratio of new to previous capacity
s = scaling factor
C = previous plant cost.

The scaling factor for mining is estimated to be approximately .75. This number is derived from data collected by the U.S. Bureau of Mines[6]. The scaling factor for chemical plants is estimated to be approximately .7 [7]. The scaling factors are limited to no more than tenfold changes in plant capacity.

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7.0 STOCKPILES

7.1 RESOURCE DESCRIPTION

The primary industrial stockpile for thorium is owned by W. R. Grace and Company in Chattanooga, Tennessee. Thorium sludge is currently stock-piled at a rate of 250 tons/yr and an estimated backlog of 1500 tons of ThO_2 equivalent is contained in the ponds. The thorium sludge is reported to contain 25% ThO_2 and some uranium (assumed to be $\sim$ 1%)[1].

Government stockpiles exist at Curtis Bay, Maryland, Miamisburg, Ohio, and Fernald, Ohio. These stockpiles are primarily in the form of thorium nitrates. Approximately 800 tons of ThO_2 equivalent are stored at Curtis Bay, Maryland, and another 400 tons of ThO_2 equivalent are located in Miamisburg, Ohio. These stockpiles are assayed at approximately 46% ThO_2. Another 360 MT of sludge which contains 88% ThO_2 equivalent is stored in Fernald, Ohio.

7.2 REFINERY DESCRIPTION

Because stockpiled material is in a preprocessed state, only a refin-ing facility would be necessary to develop the resource. The conceptual stockpile refinery is located near Chattanooga, Tennessee. The refinery utilizes Grace sludge as primary feedstock. However, government stockpiles are also processed to keep the refinery operating at rated capacity.

At rated capacity, the refinery processes 600 lb/hr of sludge and op-erates 22 hours/day, 350 days/yr for 10 years. The process is assumed to recover 95% of the thorium in the sludge.

7.2.1 Refining Process

The refining process utilizes nitric acid to dissolve the sludge. The thorium-bearing acid solution is purified by solvent extraction with tributyl phosphate in kerosene. The thorium is then precipitated using oxalic acid and calcined. The refining process is illustrated in Figure 7.1.

FIGURE 7.1.   Stockpile Refining Process

FLOW STREAMS FOR STOCKPILE REFINERY

1. 600 lb/hr sludge w/^{232}Th = 132 lb/hr
2. 315 lb/hr HNO_3
3. 517 lb/hr H_2O
4. 1432 lb/hr w/^{232}Th = 132 lb/hr
5. 698 lb/hr sludge w/^{232}Th = 2.64 lb/hr (80% solids)
6. 734 lb/hr w/^{232}Th = 129.36 lb/hr
7. 587 lb/hr of 4N HNO_3
8. 1321 lb/hr w/^{232}Th = .264 lb/hr
9. 2936 lb/hr
10. 2936 lb/hr w/^{232}Th = 129.096 lb/hr
11. 2936 lb/hr of .1N HNO_3
12. 587 lb/hr
13. 3523 lb/hr
14. 1468 lb/hr of Na_2CO_3 (5%)
15. 734 lb/hr of 4N HNO_3
16. 1468 lb/hr
17. 734 lb/hr of HNO_3 to be used to make No. 7 stream
18. 2936 lb/hr w/^{232}Th = 129.096 lb/hr
19. 2324 lb/hr H_2O
20. 612 lb/hr w/^{232}Th = 129.096 lb/hr
21. 34 lb/hr HNO_3
22. 40 lb/hr H_2O
23. 52 lb/hr oxalic acid
24. 738 lb/hr w/^{232}Th = 129.096 lb/hr
25. 133.4 lb/hr H_2O
26. 494.4 lb/hr w/^{232}Th = 1.056 lb/hr
27. 143 lb/hr Na_2CO_3
28. 637.4 lb/hr w/^{232}Th = 1.056 lb/hr
29. 377 lb/hr w/^{232}Th = 128.04 lb/hr
30. 109 lb/hr H_2O
31. 268 lb/hr w/^{232}Th - 128.04 lb/hr
32. 20 lb/hr O_2
33. 47 lb/hr H_2O and 95 lb/hr CO_2 w/^{232}Th = 2.64 lb/hr
34. 2.614 lb/hr ^{232}Th
35. 47 lb/hr H_2O and 95 lb/hr CO_2 w/^{232}Th = .026 lb/hr
36. 144 lb/hr w/^{232}Th = 125.4 lb/hr

FIGURE 7.1. (cont'd)

7.2.1.1 Feed Preparation

Sludge is dissolved in acid to obtain a solution containing 200 g/ℓ ThO_2. Metallic iron is then added and allowed to dissolve slowly to hinder formation of thorium phosphate complexes. When the addition of iron is complete, nitric acid is again introduced to achieve a concentration of 4 M free nitric acid. Large particles of gangue material are filtered off and the feed enters the solvent extraction unit[2].

7.2.1.2 Solvent Extraction

Thorium is extracted from the acid solution using an organic solvent that contains a 50% solution of tributyl phosphate (TBP) in kerosene. The leach liquor and organic solvent are brought into contact in a 10 stage mixer-settler cascade. The aqueous phase contacts the organic, loses uranium and thorium, and is discharged as raffinate. The loaded solvent is then scrubbed with nitric acid and pumped to the salt stripping process.

7.2.1.3 Stripping

The thorium solution is contacted with a .02 M solution of nitric acid. The thorium is removed, then the freed organic is passed on for recovery of uranium. The thorium-bearing aqueous stream is discharged with a concentration of 50 g Th/ℓ[2].

7.2.1.4 Uranium Recovery

The solvent phase is treated with sodium carbonate to remove the uranium, reacidified, and recycled. The uranium-bearing solution is sent to tails.

7.2.1.5 Concentration

A primary and secondary evaporative system is used to concentrate the thorium nitrate solution. The feed stream enters a secondary evaporative system which consists of three boildown tanks to concentrate the solution to about 50% of the original volume. The stream is further concentrated in a continuous single effect evaporator to a concentration of 2 lb Th/gal.[3].

7.2.1.6 Precipitation

The thorium nitrate solution is mixed with nitric acid and heated to 45°C with steam. Oxalic acid is added and the solution is agitated for several minutes to facilitate precipitation of thorium oxalate[3].

7.2.1.7 Filtration

The slurry is filtered in a rotary vacuum filter. An oxalate filter cake is discharged containing 30% to 35% water by weight. The filtrate is pumped to a plate and frame filter press for further clarification and then neutralized with a solution of sodium carbonate. The spent filtrate combines with the raffinate from the solvent extraction process prior to liquid waste treatment[3].

7.2.1.8 Drying

The oxalate cake is dried in a twin-screw dryer to a 10% moisture content to avoid spattering in the calciner. The dryer has a jacket temperature of 120°C and a screw temperature of 154°C. The thorium oxalate is discharged into a variable speed auger[4] and has a bulk density of approximately 45 lb/ft^3[3].

7.2.1.9 Calcination

The calcination is carried out in an externally fired rotary kiln for two hours. Gas flowing countercurrently exits at a temperature of 820°C. The thorium oxide product is a light, finely divided powder[4].

7.2.1.10 Packaging

The thorium oxide is packaged in 55 gallon drums for storage or shipment.

7.2.2 Effluent Control

Dust generated during drying and packaging is controlled using a 6000 cfm bag and HEPA filter system. Tailings pond beach dust is controlled using spray equipment to wet down beach material.

Liquid waste consists of discharge from the extraction and filtration steps. These wastes are transported to a clay-lined pond which is sized to allow both storage of tails and evaporation of liquid discharge. A pond

size of 4 acres was chosen. A 2.5 acre pond is also included for undissolved waste sludge from the feed preparation step. Ultimately, the tails ponds will be covered with soil and vegetated.

7.3 STOCKPILED THORIUM PRODUCTION COST

The production costs for developing ThO_2 from the stockpiles are estimated at \$5.00/lb ThO_2 for the Chattanooga Sludge, \$3.30/lb ThO_2 for the Curtis Bay sludge, and \$2.30/lb ThO_2 for the Fernald sludge.

After taking into account uncertainties in refinery capital and operating costs, thorium recovery efficiency, plant life, and plant capacity using the method described in Section 16, unit costs of \$5.80 for the Grace sludge, \$4.06 for the Curtis Bay sludge, and \$2.80 for the Fernald sludge were calculated. The costs from Table 7.1 are developed in the following sections.

TABLE 7.1. Stockpile Production Costs

Stockpile	% ThO_2	Unit Cost* of Refining $/lb ThO_2	Most likely** Unit Cost of Refining $/lb ThO_2
Chattanooga Sludge	25%	$5.00	$5.80
Curtis Bay -- Miamisburg Sludge	46%	$3.30	$4.00
Fernald Sludge	88%	$2.30	$2.80

* Under reference assumptions

** Includes effect of uncertainties

7.3.1 Refinery Production Costs

Annual production costs for refining the various sludges are summarized in Tables 7.2, 7.3, and 7.4. A fixed charge rate of 0.2668 is applied to the depreciable capital investment to convert it to an equivalent annualized cost. Similarly, a fixed charge of 10% is applied to land and working capital. Annual ThO_2 production is based on the plant running at rated capacity with the given sludge as input.

TABLE 7.2. Stockpile Refinery Production Costs: Chattanooga Sludge (25% ThO_2)

Annual Fixed Charges:

Depreciable Capital	$ 814,000
Non-Depreciable Capital	$ 32,000
Annual Operating Cost	$4,653,000
Annual Production Cost	$5,499,000
Annual ThO_2 Production	1,097,000 lb
Unit Cost of Refining	$5.00/lb ThO_2

TABLE 7.3. Stockpile Refinery Production Costs: Curtis Bay and Miamisburg (46% ThO_2)

Annual Fixed Charges:

Depreciable Capital	$ 814,000
Non-Depreciable Capital	$ 32,000
Annual Operating Costs	$5,866,000
Annual Production Cost	$6,712,000
Annual ThO_2 Production	2,019,000 lb
Unit Cost of Refining	$3.30/lb ThO_2

TABLE 7.4. Stockpiles Refinery Production Costs: Fernald (88% ThO_2)

Annual Fixed Charges on Capital Investments

Depreciable Assets	$ 814,000
Non-Depreciable Assets	$ 32,000
Annual Operating Costs	$7,887,000
Annual Production Cost	$8,733,000
Annual ThO_2 Production	3,869,000 lb
Unit Cost of Refining	$2.30/lb ThO_2

7.3.2 Refinery Capital Cost

The total capital investment for the 600 lb/hr sludge refinery is esti-
mated to be $3,370,000 (Table 7.5). The refinery and effluent control build-
ing and equipment costs were estimated using the procedure described in
Section 6.0. (See Appendix A for Table A-1.1 and Table A-1.2.)

TABLE 7.5. Stockpiles Refinery Capital Cost

Refinery Buildings and Equipment	$ 1,858,000
Effluent Control Buildings & Equipment	122,000
Access Road	100,000
Feasibility Study	100,000
Environmental Impact	100,000
Spare Parts Inventory	53,000
Contingency	322,000
Tailings Pond	238,000
Waste Sludge Landfill	157,000
TOTAL Depreciable Capital Investment	3,050,000
Total Land	33,000
Working Capital	287,000
TOTAL CAPITAL INVESTMENT	3,370,000

7.3.3 Refinery Operating Costs

Estimated refinery operating costs range from $4,653,000 to $7,887,000 depending on ThO_2 content (Table 7.6). Maintenance supplies, plant supplies, utilities, and overheads were estimated using the procedure described in Section 6.0. Utility and reagent expenses are shown in Table 7.7 for various input sludges.

7.3.4 Transportation

Transportation charges must be included for shipping sludge to refinery. The haul distance for the Chattanooga sludge (25% ThO_2) is assumed to be 20 miles. One 10 ton haul truck would move about 6 tons/day of this sludge and only one trip a day would be necessary. The capital cost of the truck is estimated to be $77,000. The fixed charge rate, assuming a life of 5 years, is approximately 0.3681 (Table 7.8). Operating labor includes costs for a driver and an individual to load and unload the sludge. Labor overhead is assumed to be approximately 35% of the cost for operating labor. Maintenance is 6% of the final capital investment. The fuel costs are calculated assuming 60 miles of travel per day at 8 mpg and a gasoline cost of $.75/gallon.

7.8

TABLE 7.6. Stockpiles Refinery Operating Costs

Fixed Operating Costs	
Direct Costs	
Operating Labor (18 men/shift)	1,219,672
Supervision	183,000
Maintenance and Repairs	195,000
Operating Supplies	29,000
Laboratory Charges	183,000
Total Direct Costs	1,197,000
Indirect Costs	
Plant Overhead	958,000
Administrative Costs	239,000
Total Indirect Costs	1,197,000
Total Fixed Operating Costs	3,006,672
Variable Operating Costs	
25% ThO_2	
Reagents	1,192,000
Utilities	409,000
Transportation	45,000
Subtotal	1,646,000
46% ThO_2	
Reagents	2,074,000
Utilities	554,000
Transportation	231,000
Subtotal	2,859,000
88% ThO_2	
Reagents	3,862,000
Utilities	787,000
Transportation	231,000
Subtotal	4,880,000
Total Operating Costs	
25% ThO_2	4,653,000
46% ThO_2	5,866,000
88% ThO_2	7,887,000

Transportation charges for Curtis Bay, Miamisburg, and Fernald stock-
piles consist of expenses of shipping sludge to the refinery in Tennessee.
Thus, a shipping distance of 900 miles round trip shall be assumed with an
assumed cost of $5.00/100 wt. Total transportation charges for shipping
2,310 tons annually are approximately $231,000.

TABLE 7.7. Stockpiled Thorium Reagent and Utilities Cost

Reagents	Unit Cost	Annual Cost (25% ThO_2)	Annual Cost (46% ThO_2)	Annual Cost (88% ThO_2)
TBP	$0.88/lb	$ 102,000	$ 102,000	$ 102,000
Kerosene	$0.40/gal	8,000	8,000	8,000
Oxalic Acid	$0.40/lb	468,000	936,000	1,790,610
Nitric Acid	$10.50/100 lb	578,000	966,000	1,848,000
Sodium Carbonate	$60/2000 lb	30,000	56,000	107,130
Ammonia	$120/2000 lb	6,000	6,000	6,000
		$1,192,000	$2,074,000	$3,861,740

Utilities				
Fuel Oil	$0.50/gal	$ 108,000	$ 253,000	$ 484,000
Electricity	$0.03/kWh	300,000	300,000	300,000
Water	$0.05/1000 gal	1,000	1,500	2,900
		$ 409,000	$ 554,500	$ 786,900

TABLE 7.8. Stockpiles Annual Transportation
Cost: 25% ThO_2

Fixed Charge	$28,000
Operating Labor	8,000
Labor Overhead	2,800
Maintenance	4,600
Fuel	2,000
	$45,400

7.4 MAJOR UNCERTAINTIES

Major areas of uncertainty affecting the stockpile cost estimates include refinery capital cost, refinery operating cost, thorium recovery efficiency, plant life, and plant capacity. Our estimates for the range of uncertainty in these areas are:

refinery capital cost	+50% to -10% of calculated value
refinery operating cost	+50% to -20% of calculated value
thorium recovery efficiency	99% to 70% with most likely value of 95%
plant life	10 years to 20 years with most likely value of 10 years
plant capacity	600 lb/hr to 100 lb/hr with most likely value of 600 lb/hr

Table 7.9 shows the impact of these uncertainty ranges on the stockpile unit cost for the three ThO_2 concentrations. From the values of the incremental unit costs, a Monte Carlo statistical analysis is performed using the procedure described in Section 16. From this analysis a most likely value and overall cost range is predicted for each sludge concentration. These results are shown below:

Ore Concentration	Most Likely Value	Range
88%	$2.80	$2.02-$4.24
46%	$4.06	$2.86-$5.58
25%	$5.80	$4.30-$8.50

TABLE 7.9. Uncertainty Ranges for Factors Affecting Stockpile Cost Estimates

7.12

	25% ThO_2		46% ThO_2		88% ThO_2	
	Most Likely Value	Range	Most Likely Value	Range	Most Likely Value	Range
Refinery Capital Cost ($/$10^3$/Year)	846	126 to 75.6	846	126 to 75.6	846	126 to 75.6
Unit Capital Cost ($/lb ThO_2)	.77	1.16 to .69	.42	.63 to .38	.22	.33 to .20
Incremental Unit Cost ($/lb ThO_2)		+.39 to -.08		+.21 to -.04		+.11 to -.02
Refinery Operating Cost 10^3/Year)	4,653	6,980 to 3,722	5,866	8,799 to 4,693	7,887	11,831 to 6,304
Unit Operating Cost ($/lb ThO_2)	4.24	6.39 to 3.39	2.91	4.36 to 2.32	2.04	3.06 to 1.63
Incremental Unit Cost ($/lb ThO_2)		+2.12 to -.85		+1.45 to -.59		+1.02 to -.41
Thorium Recovery Efficiency (%)	95	70 to 99	95	70 to 99	95	70 to 99
Unit Recovery Efficiency ($/lb ThO_2)	5.01	6.80 to 4.81	3.32	4.51 to 3.19	2.26	3.07 to 2.17
Incremental Unit Cost ($/lb ThO_2)		+1.79 to -.20		+1.19 to -.13		+.81 to -.09
Plant Life (Years)	10	10 to 20	10	10 to 20	10	10 to 20
Unit Plant Life Cost ($/lb ThO_2)	5.01	5.01 to 4.90	3.32	3.32 to 3.26	2.26	2.26 to 2.23
Incremental Unit Cost ($/lb ThO_2)		0 to -.11		0 to -.06		0 to -.03
Plant Capacity (lb/hr)	600	600 to 100	600	600 to 100	600	600 to 100
Unit Plant Capacity Cost ($/lb ThO_2)	5.01	8.57 to 5.01	3.32	5.70 to 3.32	2.26	4.24 to 2.26
Incremental Unit Cost ($/lb ThO_2)		+3.56 to 0		+2.38 to 0		+1.98 to 0

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8.0 LEMHI PASS DEPOSIT

8.1 RESOURCE DESCRIPTION

The thorium-bearing vein deposits in the Lemhi Pass area have been described in literature by Staatz[1,2]. Lying astride the Continental Divide, the Lemhi Pass quadrangle occupies approximately 53 square miles in the east-central part of the Beaverhead Mountains of Idaho and Montana. Although the Lemhi Pass thorium district is much larger, the quadrangle contains the most abundant thorium veins. Veins containing copper but no thorium, and barren quartz veins also exist. Staatz[3] estimates the thorium reserves at 67,700 tons ThO_2 with an additional 120,000 tons of probable and potential resources. Ten veins within the district contain 94% of these reserves.

The thorium-bearing veins range from "paper thin" to 30 ft in thickness. Length varies from a few feet to 3,900 ft. The largest vein, the Last Chance Vein, spans the 3,900 ft and varies in thickness from 7 ft to 30 ft. The veins strike either N 40°-50° W or N 70°-80° W and dip steeply (60°+) to the southwest[1].

The thorium content of some of the better explored thorium veins range from 0.001% to 16.3% with an average ThO_2 content of 0.4%. Grade varies considerably within the veins. Although the overall rare earths content is about equal to the thorium content, the total rare earths oxide content ranges from .042% to 10.95%.

Thirty-nine minerals have been identified from various veins, but only seven minerals -- quartz, magnetite, specularite, geothite, thorite, microcline and rutile -- occur in at least two-thirds of the veins. Other less common minerals include apatite, muscovite, monazite, red earthy hematite, black maganese oxide minerals, pyrite, and barite. The thorium-bearing veins consist mainly of quartz and quartz-microcline gangue. Veinlets of geothite and hematite cut the thorium veins and contain the thorium-bearing minerals thorite and, to a lesser degree, monazite, and in some cases, brockite and allanite. Of these, thorite is the most abundant mineral.

8.2 MINE DESCRIPTION

Lemhi Pass is not currently being mined for thorite; thus, the Lemhi Pass thorite mine is a conceptual model based on information about the deposit. The Lemhi Pass mine will utilize underground mining. The methods most likely to be used in producing the thorium vein deposits include: 1) resue stoping, and 2) cut and fill stoping depending on vein dip, vein width, and rock strength.

The referenced thorite mine has an average capacity of 1,000 tpd of ore. It operates 2 shifts/day, 250 days/yr for 28 years. During the first 14 years, approximately 2,734,000 tons of ore are mined containing approximately 14,200 tons of ThO_2. Due to inefficiencies in the mining methods, only 70% or 9,800 tons of ThO_2 are recovered. Due to dilution during mining, the average ore grade is reduced to .28%.

Over the last 14 years of mine life, an additional capital expenditure allows an additional 3,417,000 tons of ore to be accessed. Assuming a mining efficiency of 60%, approximately 13,993 tons of ThO_2 are recovered. Mean ore grade varies from .40% to 51% (Table 11, Appendix D).

Shafts may be required to mine the remainder of the reserves. Due to insufficient data, the cost for recovering the remaining reserves was not determined in this study.

8.2.1 Mining Process

The main steps in all underground mining include drilling and blasting the ore, loading and transporting ore to the surface, and establishing ground support. The following sections discuss the major process steps.

8.2.1.1 Drilling and Blasting

Blast holes are wet drilled into the working face using pneumatic drills, loaded with explosives and detonated. Detonation is usually at the end of each shift when miners are out of the work areas.

8.2.1.2 Loading and Transporting Rock to Surface

At the beginning of the following shift, broken rock is removed from the working face. A slusher draws the broken rock into a chute. The rock

falls down the chute and is loaded by gravity feed into ore cars positioned on the haulage level. When the working face and ore haulage are on the same level, the ore is loaded into the ore cars using a mucking machine. The loaded ore cars are pulled by locomotive to a dump station outside the mine.

8.2.1.3 Ground Support

After removing broken rock from the working area, support is provided, if necessary, by either installing rock bolts, split sets, or, in the case of weak ground, timber sets. Unmined pillars of waste rock or low grade ore are often left to provide support also. Depending on the mining method, waste rock or sand fill may be brought into mined out areas for additional support.

8.2.1.4 Loading and Scanning of Rock

Ore is loaded from the skips into dump trucks on the mine surface. The dump trucks are driven through a scanner to determine ore grade. The trucks are then routed to the mine storage pad. Periodically the ore is loaded from mine storage and transported to the mill.

8.2.2 Effluent Control

In working areas with high water inflow, mine water must be pumped from the mining facility. A main pump and several auxiliary pumps transport water from mine excavation points to a 2 acre holding pond on the mine surface.

Airborne effluents consist of fugitive dust, Rn-220 gas, and combustion gases. Dust suppression in working areas of the mine is accomplished by wetting down excavated rock prior to loading. Loaded trucks and ore cars along with haulage rods are similarly wetted down.

A 300,000 cfm mine ventilation system also aids in controlling airborne effluents. Mine ventilation is accomplished by installing fans and mine openings that provide air passage in and out of working areas. Auxiliary ventilation is provided in deadheaded working areas, especially in regions of high ambient temperature or high thorium concentration.

8.3 MILL DESCRIPTION

Thorite has not been successfully milled on a large scale in the U.S.; therefore, the Lemhi Pass thorite mill is based on a laboratory scale process developed by the U.S. Bureau of Mines[4]. The reference mill has a design capacity of 1,000 tpd. The process is assumed to recover 91% of the thorium and produce a ThO_2 product of 99+% purity.

The mill operates 22 hr/day, 250 days/year for 28 years. The mill is located approximately 25 miles from the mine in a flat area suitable for an evaporation pond.

8.3.1 Milling Process

The milling process involves acid leaching of thorite followed by solvent extraction and is similar to techniques used in uranium ore processing. The milling process is illustrated in Figure 8.1. The following sections discuss major process steps.

8.3.1.1 Run-of-Mine Storage

Ore arrives at the mill by truck and is stored in piles according to grade. The ore is blended before processing to provide a constant head grade for the mill. Front end loaders transfer the ore from the storage areas to conveyor. Ore is processed at a rate of 91 tons per hour.

8.3.1.2 Crushing and Grinding

The conveyor discharges the ore into a jaw crusher which is in series with a cone crusher. The minus 1 in. discharge from the cone crusher is transferred to the temporary storage bins. From there, the crushed ore is ground to -35 mesh in two 8 ft x 8 ft ball mills. After grinding, the ore is mixed with water and transferred to the leaching tanks.

8.3.1.3 Leaching

Slurry from the grinding circuit is fed directly into the first of a series of twelve 3,500 gallon stainless steel tanks equipped with steam heaters to maintain a temperature of 80°C. The slurry is mixed with sulfuric acid and agitated. The residence time is about 2 hr.[4]. The sulfuric acid

THORITE MILL / REFINERY

ORE
↓ 1

```
          ┌──────────────────┐
          │    UNLOADING     │─────────┐
          └──────────────────┘         │
                   ↓                    ├───→ 2 DUST
          ┌──────────────────┐         │
          │    CRUSHING       │─────────┘
          │   AND GRINDING    │
          └──────────────────┘
```

H₂O 3 ──────────────→   ↓ 4

H₂SO₄ 5 ──────────→ ┌──────────────────┐
                      │   LEACHING   │
                      └──────────────────┘
H₂O   ↓ 6
(FLOCCULANT) 7 ────→ ┌──────────────────┐
                       │   COUNTER   │
                       │   CURRENT   │────→ 9 TAILS
WASH   │   DECANTATION   │
WATER 8 ───────────→ └──────────────────┘
                               ↓ 10
                       ┌──────────────────┐
                       │   AMINE   │────→ 11 TAILS
                       │   EXTRACTION   │
                       └──────────────────┘
                          ↓ 12   ↑ 13
STRIP   ┌──────────────────┐
SOL'N 14 ──────────→ │   SALT STRIPPING │
                       └──────────────────┘
                               ↓ 15
Na₂CO₃ 16 ─────────→ ┌──────────────────┐
                       │ REDUCTION AND   │
Na₂S₂O₄ 17 ────────→ │ NEUTRALIZATION   │
                       └──────────────────┘
                               ↓ 18
WASH   ┌──────────────────┐
SOL'N 19 ──────────→ │   DEWATERING   │────→ 20 TAILS
                       └──────────────────┘
                               ↓ 21
NaOH 22 ───────────→ ┌──────────────────┐
                       │   CAUSTIC   │
                       │   DIGESTION   │
                       └──────────────────┘
                               ↓ 23
WASH 24 ───────────→ ┌──────────────────┐
H₂O   │   FILTRATION   │────→ 25 TAILS
                       └──────────────────┘
                               ↓ 26
                       ┌──────────────────┐
                       │   DRYING AND   │────→ 27 H₂O+DUST
                       │   IGNITION   │
                       └──────────────────┘
                               ↓ 28
                       ┌──────────────────┐
                       │   PACKAGING   │────→ 29 DUST
                       └──────────────────┘
                               ↓
                              30
```

FIGURE 8.1.  Lemhi Pass Milling/Refining Process

# FLOW STREAMS FOR LEMHI PASS MILL/REFINERY

1. 45.5 tons/hr ore w/$^{232}$Th = 285.38 lb/hr

2. 4.55 lb/hr w/$^{232}$Th = .0132 lb/hr

3. 18.2 tons/hr $H_2O$ or 72.8 gpm

4. 63.7 tons/hr w/$^{232}$Th = 285.37 lb/hr

5. 16,835 lb/hr $H_2SO_4$

6. 72.12 tons/hr w/$^{232}$Th = 285.37 lb/hr

7. 4.55 tons/hr of $H_2O$ (flocculant)

8. 168.35 tons/hr of wash $H_2O$

9. 45% solids - 63.39 tons/hr solids w/$^{232}$Th = 20.86 lb/hr

   55% solution - 77.475 tons/hr solution w/$^{232}$Th = 2.45 lb/hr

10. 104.155 tons/hr w/$^{232}$Th = 262.06 lb/hr

11. 104.155 tons/hr $^{232}$Th = 1.64 lb/hr

12. 19.5 tons/hr w/$^{232}$Th = 260.42 lb/hr

13. 19.5 tons/hr of stripped organic

14. 12.7 tons/hr of strip solution

15. 12.7 tons/hr w/$^{232}$Th = 260.42 lb/hr

16. 590.5 lb/hr $Na_2CO_3$

17. 9.35 lb/hr $Na_2S_2O_4$

18. 13 tons/hr w/$^{232}$Th = 260.42 lb/hr

19. 3.16 tons/hr of wash solution (3% $Na_2SO_4$)

20. 12.49 tons/hr w/$^{232}$Th = .55 lb/hr

21. 3.67 tons/hr w/$^{232}$Th = 259.87 lb/hr

22. 95.5 lb/hr of NaOH

23. 3.72 tons/hr w/$^{232}$Th = 259.87 lb/hr

24. 6.25 tons/hr of wash $H_2O$

25. 6.25 tons/hr w/$^{232}$Th = .0273 lb/hr

26. .597 ton/hr w/$^{232}$Th = 259.84 lb/hr

27. 896.2 lb/hr s/$^{232}$Th = .105 lb/hr

28. 297.8 lb/hr w/$^{232}$Th = 295.735 lb/hr

29. .021 lb/hr w/$^{232}$Th = .0185 lb/hr

30. 297.8 lb/hr w/$^{232}$Th = 259.72

FIGURE 8.1. (cont'd)

dissolves the thorium along with impurities in the ore. The sulfuric acid consumption, which is a function of the ore grade and the soluble impurities, can vary from 370 lb/ton to 650 lb/ton of ore.

### 8.3.1.4 Countercurrent Decantation

The leach liquor contains approximately 60% solids which must be removed before the solvent extraction. This is accomplished in a five-stage countercurrent decantation process. Flocculant is added to increase the settling rate of the solids. The underflow from the fifth thickener goes to the tailings pond. Based on a two hour settling rate, a settling area of 22 $ft^2$/ton/day is required for the thickener. Final clarification is accomplished using a leaf-type pressure filter with a vacuum of 15 inches of mercury.[4]

### 8.3.1.5 Amine Extraction

Thorium is extracted from the leach liquor by an organic solvent containing 10% primary amine, 5% primary decyl alcohol, and 85% kerosene. The leach liquor and organic solvent are brought into contact in seven 2,500 gallon mixer/settlers for 15 minutes. The organic losses are estimated to be 1/2 gallon/ton processed per hour[4]. After the thorium is extracted from the leach liquor, the leach liquor is sent to the tails pond and the thorium containing organic is pumped to the salt stripping process.

### 8.3.1.6 Salt Stripping

The thorium is stripped from the organic using a premixed salt solution containing 1.5 M NaCl and 0.5 M HCl. The organic and the salt solution are brought into contact in stainless steel agitated tanks. The salt solution removes the thorium from the organic and the organic is returned to the extraction step. The thorium containing salt solution is pumped to the reduction and neutralization process.

### 8.3.1.7 Reduction and Neutralization

The salt solution is neutralized with soda ash in agitated tanks to remove the thorium, and sodium hydrosulfide is added to reduce ferric iron[4]. The precipitate is dewatered using a disk centrifuge. The filtrate is discarded to the tailings pond.

### 8.3.1.8 Caustic Digestion

The dewatered precipitate is pumped to agitated tanks where the precipitate is digested with caustic soda at ambient temperature. A two hour resident time is required and approximately 3 lb of sodium hydroxide are used for each ton of ore[4]. The caustic digestion forms thorium hydroxide which is removed from the caustic solution by filtration with a continuous rotary drum filter. The filter has approximately 4 ft$^2$ of filter area per ton/hr and yields a precipitate containing 70% solids. The filtrate is pumped to the tails pond and the cake is discharged to a belt conveyor.

### 8.3.1.9 Drying and Ignition

The hydroxide cake is dried to a moisture content of 10% in a twin-screw dryer with a jacket temperature of 120°C and a screw temperature of 154°C. The twin screw dryer is discharged to a rotary kiln where the thorium hydroxide is converted to thorium oxide. The kiln operates at 900°C[4]. The kiln was sized by assuming that, in normal operation, 7% of the kiln volume is filled with thorium hydroxide. The rotary kiln discharges ThO$_2$ of 99+% purity, which is packaged in drums for shipment to the fuel fabrication facility.

## 8.3.2 Effluent Control

Two types of effluent are discharged from the mill. Airborne wastes are generated in the crushing, grinding, drying and bagging operation. In addition, liquid and slurry wastes are discharged at various steps in the milling process.

A 26,000 cfm cyclone dust collection system is used for reducing radioactive dust and silica dust in the unloading, crushing and grinding circuits[5]. Dust from drying and ignition steps is controlled in a 21,000 cfm jet impingement scrubber. Packaged dust is sent to a 4000 cfm bag filter system. Beach dust from the tailings pond is controlled by wetting down with portable spray equipment.

Liquid waste consists of discharge from the leaching, countercurrent decantation, amine extraction, dewatering and filtration processes. These

waste streams are transported to a tails/evaporation pond which is sized to allow both storage of tails and evaporation of liquid discharge. A pond size of 100 acres was chosen based on a net evaporation of 18"/year and recycle of 3/4 of the process water. Ultimately, the tails pond will be covered with soil and vegetated.

### 8.3.3 Sulfuric Acid Plant

In order to reduce the cost of sulfuric acid, an onsite sulfuric acid plant is included with the thorite mill.

### 8.4 LEMHI PASS PRODUCTION COST

The production costs for developing an estimated 23,754 tons of $ThO_2$ from Lemhi Pass are estimated at $14.60/lb $ThO_2$ for the first 14 years and $11.30/lb $ThO_2$ for the next 14 years. After taking into account uncertainties in mine capital and operating costs, mill capital and operating costs, mill capacity, and mill recovery efficiency, a unit cost of $15.73/lb for the first stage and $12.10/lb for the second stage were found by the method described in Section 16. The costs from Table 8.1 are developed in the following sections.

TABLE 8.1. Lemhi Pass Production Costs

|  | Unit Cost ($/lb $ThO_2$) | |
|---|---|---|
|  | Stage 1 | Stage 2 |
| Mining | $ 4.00 | $ 3.86 |
| Milling | $10.60 | $ 7.40 |
| TOTAL from estimates | $14.60 | $11.30 |
| TOTAL including uncertainty (See Section 8.5) | $15.70 | $12.10 |

### 8.4.1 Mine Production Costs

The production costs for mining the Lemhi Pass resource are shown in Table 8.2.

TABLE 8.2. Lemhi Pass Mine Production Costs

|                                               | Stage 1         | Stage 2        |
|-----------------------------------------------|-----------------|----------------|
| Annualized Capital Cost                       | $ 1,878,000     | $   981,000    |
| Annual Operating Costs                        | $ 4,556,000     | $ 7,904,000    |
| Subtotal                                      | $ 6,434,000     | $ 8,885,000    |
| Depletion Allowance                           | $ 1,415,000     | $ 1,954,000    |
| Annual Production Cost                        | $ 5,019,000     | $ 6,931,000    |
| Annual Levelized Production of $ThO_2$        | 1,254,000 lb    | 1,797,000 lb   |
| Unit Cost of Mining                           | $4.00/lb        | $3.90/lb       |

Mine capital costs are developed in two stages for the: (1) 9,761 tons of $ThO_2$ and (2) 13,993 tons of $ThO_2$ recovered at Lemhi Pass. The first stage requires a capital expenditure of $7,611,500 plus a charge of $35,200 for a settling pond to handle mine water and $45,000 for miscellaneous equipment to handle uncontrolled dust from storage piles and roads. After a period of 14 years an additional $4,020,000 are expended to enable the mine to recover tonnage for another 14 years. A fixed charge rate of 0.2441 is applied to annualize the capital investments for each stage.

Mine operating costs for the first stage are approximately $4,556,000 annually. Operating costs for the second stage average $7,904,000 per year. The depletion allowance is 22% of the sum of the annualized capital cost and annual operating cost. For a more detailed description of mine design and operation, refer to Appendix D.

8.4.2 Mill Production Costs

Production costs for milling the Lemhi Pass thorite are summarized in Table 8.3. A fixed charge rate of .2336 is applied to convert the capital investment into an equivalent annual cost.

8.4.2.1 Mill Capital Cost

The total capital investment for a 1,000 ton/day Lemhi Pass thorite mill is estimated to be $21,701,000. Itemized capital costs are listed in Table 8.4. The mill capital cost and the effluent control equipment capital costs were estimated using the procedure described in Section 6.0. The

8.10

**TABLE 8.3.** Lemhi Pass Mill Production Cost

|                                   | Stage 1        | Stage 2        |
|-----------------------------------|----------------|----------------|
| Annual Fixed Charges              |                |                |
| Depreciable Capital               | $ 4,663,000    | $ 4,663,000    |
| Non-Depreciable Capital           | $   174,000    | $   174,000    |
| Annual Operating Cost             | $ 8,453,000    | $ 8,453,000    |
| Annual Production Cost            | $13,290,000    | $13,290,000    |
| Annual Levelized Production of $ThO_2$ | 1,254,000 lb | 1,797,000 lb  |
| Unit Cost of Milling              | $10.60/lb      | $ 7.40/lb      |

**TABLE 8.4.** Lemhi Pass Mill Capital Cost

| | |
|---|---|
| Mill Buildings and Equipment Cost | $ 9,917,000 |
| Effluent Control Buildings & Equipment Cost | $   142,000 |
| Sulfuric Acid Plant Buildings & Equipment Cost | $ 2,495,000 |
| Access Road | $   100,000 |
| Feasibility Study | $   100,000 |
| Environmental Impact | $   100,000 |
| Contingency | $ 3,700,000 |
| Tailings Pond (100 acre) | $ 3,406,000 |
| Total Depreciable Capital Investment | $19,960,000 |
| Total Land | $   341,000 |
| Working Capital | $ 1,400,000 |
| Total Capital Investment | $21,701,000 |

capital cost of the sulfuric acid plant was estimated from Guthrie[5] based on an acid consumption of 62,500 tons/yr (500 lb acid/ton of ore). The mill capital cost calculations, effluent control equipment capital cost, and the sulfuric acid plant capital costs are developed in Appendix A (Tables A-2.1, A-2.2, and A-2.3).

8.4.2.2 Mill Operating Cost

The annual operating costs for the Lemhi Pass thorite mill are estimated at $8,453,000 (Table 8.5). Maintenance supplies, plant supplies, utilities

8.11

and overheads were estimated using the procedure described in Section 6.0.
Mill labor costs, reagent expenses, and sulfuric acid plant operating costs
will be discussed below.

TABLE 8.5. Lemhi Pass Mill Operating Costs

Direct Cost

| | |
|---|---:|
| Operating Labor (81 men) | 1,306,000 |
| Direct Supervisory and Clerical | 196,000 |
| Maintenance and Repairs | 1,302,000 |
| Operating Supplies | 195,000 |
| Laboratory Charges | 196,000 |
| Reagents | 885,000 |
| Utilities | 950,000 |
| Sulfuric Acid Plant Operating Cost | 1,320,000 |

Indirect Cost

| | |
|---|---:|
| Plant Overhead | 1,682,000 |
| Administrative Overhead | 421,000 |
| TOTAL | $8,453,000 |

### 8.4.2.3 Mill Labor Cost

Since the solvent extraction process used in the Lemhi Pass thorite mill
is similar to that used in existing uranium mills, the operating labor require-
ments are derived from statistics on uranium ore processing. The operating
labor requirements for a 1000 ton/day uranium mill are assumed to be 12.3
tons/man-day or eighty-one men[6].

### 8.4.2.4 Reagents

Reagent expenses were calculated by estimating the annual theoretical
consumption of each reagent and multiplying by the current cost of the rea-
gent. Since a sulfuric acid plant is provided, no expense for sulfuric acid
is included. Table 8.6 presents reagent consumption, unit cost and annual
cost. The total reagent expense is $885,000.

8.12

TABLE 8.6.  Lemhi Pass Reagent Cost

| Reagent | Annual Consumption | Unit Cost[a] | Annual Cost |
|---|---|---|---|
| NaCl | 9,000 tons | $ 41/ton | $ 369,000 |
| Kerosene | 88,000 lb | 0.4/gal | 5,000 |
| HCl | 1,875 tons | 61.3/ton | 115,000 |
| $NaCO_3$ | 2,500 tons | 68.9/ton | 172,000 |
| $Na_2SO_2O_4$ | 7,700 lb | 0.58/lb | 4,000 |
| NaOH | 753,500 lb | 0.24/lb | 181,000 |
| $NaSO_4$ | 751 tons | 51.9/ton | 739,000 |
| TOTAL Reagent Expense | | | $ 885,000 |

[a] From Chemical Marketing Report, January 23, 1978.

8.4.2.4  Sulfuric Acid Plant Operating Expense

The operating expense for the sulfuric acid plant (raw materials, operating supplies, maintenance materials, labor, supervision, utilities and indirect cost) were estimated from Guthrie[5]. For a 62,500 tpy sulfuric acid plant escalated to January 1978, the operating cost was estimated to be $1,320,000/yr.

8.5  MAJOR UNCERTAINTIES

Major areas of uncertainty affecting the $ThO_2$ unit cost estimate include mine capital and operating costs, mill capital and operating costs, mill capacity, and mill recovery efficiency. Our estimates for the range of uncertainty in these areas are:

| | |
|---|---|
| Mine Capital Cost | +50% to -10% of calculated value |
| Mine Operating Cost | +50% to -10% of calculated value |
| Mill Capital Cost | +50% to -10% of calculated value |
| Mill Operating Cost | +50% to -10% of calculated value |
| Mill Capacity | 2000 tpd to 500 tpd with most likely value of 1000 tpd |
| Mill Recovery Efficiency | 86% to 48% with most likely value of 64% |

Table 8.7 shows the impact of these uncertainty ranges on the unit cost of $ThO_2$ for the two stages. From the incremental unit cost values, a Monte Carlo statistical analysis is performed using the procedure described in Section 16. A most likely value and overall cost range is predicted for each stage from this analysis and is shown below:

| Stage | Most Likely Value ($/lb $ThO_2$) | Range ($/lb $ThO_2$) |
|-------|-----------------------------------|------------------------|
| 1     | 15.73                             | $10.80 to $22.90       |
| 2     | 12.10                             | $ 8.91 to $17.40       |

TABLE 8.7. Uncertainty Ranges for Factors Affecting $ThO_2$ Unit Cost for Lemhi Pass

| | Stage 1 | | Stage 2 | |
| | Most Likely Value | Range | Most Likely Value | Range |
|---|---|---|---|---|
| Mine Capital Cost | $ 1,878,000 | $2,817,000 to $1,690,000 | $ 981,000 | $ 1,472,000 to $883,000 |
| Unit Mill Capital Cost | $1.17 | $1.75 to $1.05 | $.43 | $.64 to $.38 |
| Incremental Unit Cost | | +.58 to -.12 | | +.21 to -.05 |
| | | | | |
| Mine Operating Cost | $ 4,556,000 | $6,834,000 to $4,100,000 | $ 7,904,000 | $11,856,000 to $7,114,000 |
| Unit Mill Operating Cost | $2.83 | $4.25 to $2.55 | $3.43 | $5.15 to $3.09 |
| Incremental Unit Cost | | +$1.42 to -$.28 | | +$1.72 to -$.34 |
| | | | | |
| Mill Capital Cost | $ 4,837,000 | $7,256,000 to $4,353,000 | $ 4,837,000 | $ 7,256,000 to $4,353,000 |
| Unit Mill Capital Cost | $3.86 | $5.79 to $3.47 | $2.69 | $4.04 to $2.42 |
| Incremental Unit Cost | | +$1.93 to -$.39 | | +$1.35 to -$.27 |
| | | | | |
| Mill Operating Cost | $ 8,453,000 | $12,680,000 to $7,608,000 | $ 8,453,000 | $12,680,000 to $7,608,000 |
| Unit Mill Operating Cost | $6.74 | $10.11 to $6,07 | $4.70 | $7.06 to $4.23 |
| Incremental Unit Cost | | +3.37 to -$.67 | | +$2.36 to -$.47 |
| | | | | |
| Plant Capacity | 1000 tpd | 500-2,000 tpd | 1000 tpd | 500-2,000 tpd |
| Unit Plant Capacity Cost | $14.60 | $18.02 to $11.92 | $11.26 | $13.89 to $9.21 |
| Incremental Unit Cost | | +$3.42 to -$2.68 | | +$2.63 to -$2.05 |
| | | | | |
| Ore Recovery | 64% | 86% to 48% | 64% | 48% to 86% |
| Unit Ore Recovery Cost | $14.60 | $19.47 to $10.87 | $11.26 | $15.01 to $8.38 |
| Incremental Unit Cost | | +$4.87 to -$3.73 | | +$3.75 to -$2.88 |

# REFERENCES

1.  M. H. Staatz, Geology and Description of the Thorium-Bearing Veins, Lemhi Pass Quadrangle, Idaho and Montana. USGS Bulletin 1351, 94 pp., 1972.

2.  M. H. Staatz, "Thorium Veins in the United States". Economic Geology, 69:494-507, 1974.

3.  M. H. Staatz, Proceedings of the Uranium Industry Seminar, 1978, GJO-108(78), U.S. Dept. of Energy, Grand Junction, Colorado, pp. 109-115, 1978.

4.  S. R. Borrowman and J. B. Rosenbaum, Recovery of Thorium From Ores in Colorado, Idaho, and Montana, USBM 5916, United States Bureau of Mines, Update, 16 pp. 1976.

5.  K. M. Guthrie, Process Plant Estimating Evaluation and Control, Craftman Book Company of America, pp. 80-97, 1974.

6.  R. C. Merritt, The Extraction Metallurgy of Uranium, Colorado School of Mines Research Institute, p. 377, 1971.

# 9.0 THORIUM PRODUCTION FROM HALL MOUNTAIN, IDAHO

## 9.1 RESOURCE DESCRIPTION

Hall Mountain is located approximately 3 miles east of the U.S. border Station at Porthill, Idaho[1]. Thorium veins exist in the quartz diorite of the Purcell sills and in the belt subgroup of the Prichard Formation. The thorium-bearing mineralization, mainly thorite, is erratic and commonly occurs along fractures.

Thorium has been identified in 11 narrow, steeply dipping veins in a half-square mile area of the deposit. The veins strike from N42°W to N30°E and dip into the hill from 35° to over 60° from the horizontal. The thickness of the veins range from less than 1 ft to 13 ft with an average width of 4 ft. The exposed length of the veins range from 6 ft to 700 ft with an average length of 185 ft. The poorly exposed veins out crop between 2400 ft and 4000 ft above sea level and are covered with lake deposits and glacial till below 2400 ft.

Mineralization of the Hall Mountain area has not been quantified. However, minerals associated with thorite include quartz, calcite, chlorite, altite, and sphene. Based on geological and sample data published by Staatz[2], the mineable ore reserve is estimated at 5210 tons of $ThO_2$ with an average ore grade of 1.3%. The probable potential resource in this district is 29,900 tons. This area contains 3.7% of the total resource for identified vein deposits in the U.S.

## 9.2 MINE DESCRIPTION

Currently, Hall Mountain is not being mined for thorium. The conceptual Hall Mountain mine is thus based on the Lemhi Pass mine assuming a similar geology. The conceptual mine produces approximately 400 tpd with a mine life of 5 yr. The mine is designed to operate two shifts/day, 250 days/yr. A dilution rate of 15% is assumed for this study. Thus, the average run of mine grade is assumed to be 1.1%.

### 9.2.1 Mining Process

The process for mining Hall Mountain thorite is essentially the same as for mining Lemhi Pass thorite. The mining process is described in Section 8.2.1 of this report.

## 9.3 MILL DESCRIPTION

The reference thorite mill utilizes acid leaching of ore followed by solvent extraction. The reference thorite mill has a capacity of 400 tpd of ore which matches the output from the Hall Mountain thorite mine. The mill is assumed to operate 5 yr. Since the average thorium content of the ore is estimated to be 1.1%, the mill would produce about 8090 lb/day of $ThO_2$.

### 9.3.1 Milling Process

The process for milling Hall Mountain thorite is assumed to be the same as for milling Lemhi Pass thorite. The process is described in Section 8.3.1 of this report.

## 9.4 HALL MOUNTAIN PRODUCTION COSTS

The production costs for developing an estimated 5210 tons/$ThO_2$ from Hall Mountain is $6.00/lb $ThO_2$ (Table 9.1). After analyzing the uncertainty in mine capital and operating costs, mill capital and operating costs, plant capacity, and mill recovery efficiency, a unit cost of $6.30/lb $ThO_2$ was found by the method described in Section 16. Costs from Table 9.1 are developed in the following sections.

TABLE 9.1. Hall Mountain Production Costs

|  | Unit Cost ($/lb/$ThO_2$) |
|---|---|
| Mining | $1.41 |
| Milling | $4.61 |
| Total from Estimates | $6.00 |
| Total Including Uncertainty | $6.30 |

### 9.4.1 Mine Production Costs

The mine production costs for operating the Hall Mountain mine are shown in Table 9.2. A fixed charge rate of 0.3681 is applied to the capital investment. Capital and operating costs for the mine are developed in the following sections.

TABLE 9.2. Hall Mountain Mine Production Costs

| | |
|---|---|
| Annualized Capital Cost | $1,434,000 |
| Annual Operating Costs | 2,217,000 |
| Subtotal | $3,651,000 |
| Depletion Allowance | 803,000 |
| Annual Production Costs | $2,848,000 |
| Average Annual Production of $ThO_2$ | 2,023,000 lbs |
| Unit Cost of Mining | $1.40/lb |

#### 9.4.1.1 Mine Capital Costs

Capital costs are calculated from the Lemhi Pass cost estimate using a 0.75 scaling factor[4]. The estimated capital cost is calculated to be $3,829,000. A settling pond to handle mine water costs an additional $21,700. Miscellaneous equipment to handle uncontrolled dust from storage piles and roads costs approximately $45,000. Total capital cost, including effluent control, is $3,895,000.

#### 9.4.1.2 Mine Operating Costs

The mine operating cost is estimated using the unit cost calculated from the first 7 years of operating the Lemhi Pass mine (Table 5, Appendix D). This value is $22.17/ton of ore and includes expenses for exploration and development. Assuming a production rate of 400 tpd for 250 days/yr, the annual operating cost for Hall Mountain is $2,217,000.

### 9.4.2 Mill Production Costs

The production cost for milling Hall Mountain ore is $4.60/lb $ThO_2$ (Table 9.3). A fixed charge rate of 0.3681 is applied to the capital investment. Mill Capital and operating costs are discussed in the following sections.

TABLE 9.3. Hall Mountain Mill Production Costs

Annual Fixed Charges:

| | |
|---|---|
| Depreciable Capital | $3,869,000 |
| Non-Depreciable Capital | 92,000 |
| Annual Operating Cost | $5,363,000 |
| Production Cost | $9,324,000 |
| Annual ThO$_2$ Production | $2,023,000 |
| Unit Cost of Milling | $4.60/lb ThO$_2$ |

### 9.4.2.1 Mill Capital Costs

The total capital investment for a 400 tpd thorite mill is estimated at $11,426,000. The capital cost is scaled down from the Lemhi Pass mill capital cost by multiplying by the capacity ratio to the 0.7 power[3]. The costs are summarized in Table 9.4.

TABLE 9.4. Hall Mountain Mill Capital Cost

| | 400 Tons Per Day |
|---|---|
| Total Depreciable Capital Investment | $ 10,510,000 |
| Land | 179,000 |
| Working Capital | 737,000 |
| TOTAL Capital Investment | $ 11,426,000 |

### 9.4.2.2 Mill Operating Costs

The components of annual operating cost are shown in Table 9.5. Costs for electric power and fuel oil are scaled down using the 0.7 scaling factor[3] because they depend on equipment size and would be influenced by economies of scale. Total labor force is estimated from typical manpower requirements of a uranium mill of similar size[5]. Operating labor rates are estimated at $8.06/hr. Reagents and water consumption are scaled down linearly.

TABLE 9.5. Hall Mountain Mill Operating Cost

Direct Costs

| | |
|---|---:|
| Operating Labor (62 men) | $ 999,000 |
| Supervisory and Clerical | 150,000 |
| Maintenance and Repairs | 685,000 |
| Operating Supplies | 103,000 |
| Laboratory Charges | 150,000 |
| Reagents | 354,000 |
| Utilities | |
|     Fuel Oil | 301,000 |
|     Electricity | 217,000 |
|     Water | 29,000 |
| Sulfuric Acid Operating Cost | 1,000,000 |

Indirect Costs

| | |
|---|---:|
| Plant Overhead | 1,100,000 |
| Administrative Costs | 275,000 |
| TOTAL Operating Costs | $5,363,000 |

## 9.5 MAJOR UNCERTAINTIES

Major areas of uncertainty affecting the $ThO_2$ unit cost estimates include mine capital and operating costs, mill capital and operating costs, plant capacity, and mill recovery efficiency. Our estimates for the range of uncertainty in these areas are:

| | |
|---|---|
| Mine Capital Cost | +50% to -10% of calculated value |
| Mine Operating Cost | +50% to -10% of calculated value |
| Mill Capital Cost | +50% to -10% of calculated value |
| Mill Operating Cost | +50% to -10% of calculated value |
| Plant Capacity | 400 tpd to 200 tpd with most likely value of 400 tpd |
| Mill Recovery Efficiency | 54% to 96% with a most likely value of 91% |

The impact of these uncertainty ranges on the unit cost of $ThO_2$ is shown in Table 9.6. From the incremental unit cost values, a Monte Carlo statistical analysis is performed using the procedure described in Section 16. From this analysis a most likely value of $6.30/lb $ThO_2$ and overall cost range of $5.60/lb to $9.30/lb is predicted for the unit cost of $ThO_2$.

TABLE 9.6. Uncertainty Ranges for Factors Affecting $ThO_2$ Unit Cost for Hall Mountain

| | Most Likely Value | Range |
|---|---|---|
| Mine Capital Cost $10^3$/yr | 1,119[a] | 1,678 to 1,007 |
| Unit Mine Capital Cost $/lb $ThO_2$ | .55 | .83 to .50 |
| Incremental Unit Cost $/lb $ThO_2$ | | +.28 to -.05 |
| Mine Operating Cost $10^3$ | 1,729[a] | 2,594 to 1,556 |
| Unit Mine Operating Cost $/lb $ThO_2$ | .85 | 1.28 to .77 |
| Incremental Unit Cost $/lb $ThO_2$ | | +.43 to -.08 |
| Mill Capital Cost $10^3$ | 3,961 | 5,942 to 3,565 |
| Unit Mill Capital Cost $/lb $ThO_2$ | 1.96 | 2.94 to 1.76 |
| Incremental Unit Cost $/lb $ThO_2$ | | +.98 to -.20 |
| Mill Operating Cost $10^3$ | 5,363 | 8,045 to 4,827 |
| Unit Mill Operating Cost $/lb $ThO_2$ | 2.65 | 3.98 to 2.39 |
| Incremental Unit Cost $/lb $ThO_2$ | | +1.33 to -.26 |
| Plant Capacity tpd | 400 | 400 to 200 |
| Unit Plant Capacity Cost $/lb $ThO_2$ | | 5.40 to 6.53 |
| Incremental Unit Cost $/lb $ThO_2$ | | +1.13 to 0 |
| Thorium Recovery Efficiency % | 91 | 54% to 96% |
| Unit Recovery Efficiency Cost $/lb $ThO_2$ | 6.02 | 10.14 to 5.71 |
| Incremental Unit Cost $/lb $ThO_2$ | | +4.12 to -.31 |

(a) Depletion allowance (22%) is subtracted from cost.
(b) Mill recovery efficiency only. No data is available to predict mine efficiency.

# REFERENCES

1. M. H. Staatz, Thorium Rich Veins of Hall Mountain in Northernmost Idaho, Economic Geology, Vol. 67, 1972, pp. 240-248.

2. M. H. Staatz, Proceedings of the Uranium Industry Seminar, 1978, GJO-108 (78), U.S. Dept. of Energy, Grand Junction, Colorado, pp. 109-115, 1978.

3. Robert H. Perry, Chemical Engineers Handbook, 5th Ed., McGraw-Hill, Inc., New York, pp. 25-16 and 25-17, 1973.

4. Chemical and Operating Cost Estimating System Handbook, Prepared for the United States Dept. of Interior, Bureau of Mines, by STRAAM Engineers, Inc., Arcadia, CA, 374 pp., July 1978.

5. R. C. Merritt, The Extractive Metallurgy of Uranium, Colorado School of Mines Research Institute, Golden, CO, p. 377, 1971.

## 10.0 THORIUM PRODUCTION FROM WET MOUNTAIN

### 10.1 RESOURCE DESCRIPTION

Thorium has been identified in over 400 narrow, steeply dipping veins in a 22 square mile tract of precambrian rocks on the flank of the Wet Mountain in Colorado[1,2,3].

These veins range in thickness from less than 1 ft to 50 ft with an average thickness of about 5 ft and dip in excess of 60° from horizontal. The topography is rugged, ranging from 7,900 ft to 9,500 ft above sea level, and the veins are poorly exposed with the exposed length ranging from 1 ft to 5,000 ft.

The thorium-bearing mineralization has been described as a "thorite like mineral"[4], and is presumed to be thorite. The economic thorium vein minerals occur as irregularly distributed constituents of veins, as disseminations, and as small local concentrations in border areas of shattered rock. The $ThO_2$ content of these vein deposits ranges from 0.02% to 12.5% with the average ore grade of 1.1%.

Staatz has estimated the size of these reserves at 66,900 tons of $ThO_2$. Total probable potential resource in this district is 158,000 tons, or 46.9% of the total reserve for identified vein deposits in the U.S.[4]

### 10.2 MINE DESCRIPTION

Currently, Wet Mountain is not being mined for $ThO_2$. Thus, the Wet Mountain mine is modeled after the Lemhi Pass mine. Because of the scattered nature of the deposits, several small mines will be utilized to recover the tonnage rather than one large mine. The conceptual mine produces 350 tpd of ore and operates for 5 years. The mine runs 2 shifts/day for 250 days/yr. A dilution rate of 15% is assumed and thus the run of mine grade is reduced to .96%.

#### 10.2.1 Mining Process

The mining process is essentially the same as for Lemhi Pass thorite. This process is described in Section 8.2.1 of this report.

## 10.3 MILL DESCRIPTION

The reference thorite mill utilizes acid leaching of ore followed by solvent extraction. The mill has a capacity of 350 tpd of ore and operates for a life of 5 yr at 3 shifts/day for 250 days/yr. The mill is assumed to process ore with an average grade of 0.96% and produces 99+% $ThO_2$ with 91% recovery, or approximately 6,120 tpd of reactor grade $ThO_2$.

### 10.3.1 Milling Process

The process for milling Wet Mountain thorite is essentially the same as the process for milling Lemhi Pass thorite. This process is described in Section 8.3.1 of this report.

## 10.4 WET MOUNTAIN PRODUCTION COSTS

The production costs for developing an estimated 66,900 tons $ThO_2$ from Wet Mountain is $7.30/lb $ThO_2$ (Table 10.1). After analyzing the uncertainty in mine capital and operating costs, mill capital and operating costs, mine capacity, mill capacity, and mill recovery efficiency. a unit cost of $8.30/lb $ThO_2$ was found by the method described in Section 16. Costs from Table 10.1 are developed in the following sections.

### TABLE 10.1  Wet Mountain Production Costs

|  | Unit Cost $/lb $ThO_2$ |
| --- | --- |
| Mining | $1.83 |
| Milling | $5.48 |
| TOTAL from estimates | $7.30 |
| TOTAL (including uncertainty) | $8.30 |

### 10.4.1 Mine Production Costs

The mine production costs for Wet Mountain are shown in Table 10.2. The unit cost of mining $ThO_2$ is $1.80/lb $ThO_2$. A fixed charge rate of 0.3681 is applied to the capital investment to annualize it over five years. Capital and operating costs are developed in the following sections.

10.2

**TABLE 10.2.** Wet Mountain Mine Production Costs

| | |
|---|---|
| Annualized Capital Cost | $1,643,000 |
| Annual Operating Costs | 1,940,000 |
| Subtotal | $3,583,000 |
| Depletion Allowance | 788,000 |
| Annual Production Cost | $2,795,000 |
| Annual Production of $ThO_2$ | 1,528,800 lb |
| Unit Cost of Mining | $1.80/lb $ThO_2$ |

### 10.4.1.1 Mine Capital Cost

The capital cost for the Wet Mountain mine is calculated by scaling down the Lemhi Pass cost estimate with a .75 scaling factor[5]. Using this approximation, the capital cost is $3,463,000. An additional $1,000,000 is added to account for additional exploration necessary due to the scattered nature of this deposit and $45,700 is added for effluent control equipment bringing the cost to $4,508,700.

### 10.4.1.2 Mine Operating Cost

Operating costs are estimated to be the same as those from the first 7 years of the Lemhi Pass model (Table 5, Appendix D). Multiplying the value of $22.17/ton of ore by a production rate of 350 tpd, operating costs are calculated to be $1,940,000 annually.

### 10.4.2 Mill Production Costs

The production costs for milling Wet Mountain thorite is $5.50/lb $ThO_2$. (Table 10.3). The cost assumes a fixed charge rate of .3681 applied to the capital investment. Mill capital and operating costs are described in the following sections.

### 10.4.2.1 Mill Capital Costs

The total capital investment for a 350 tpd thorite mill is estimated at $10,406,000. The capital cost is determined by scaling down the Lemhi Pass thorite mill with a 0.70 scaling factor. Capital costs for Wet Mountain mill are summarized in Table 10.4.

## TABLE 10.3. Wet Mountain Mill Production Costs

Annual Fixed Charges:

| | |
|---|---|
| Depreciable Capital | $3,523,000 |
| Non-Depreciable Capital | $ 83,000 |
| Annual Operating Cost | $4,763,000 |
| Annual Production Cost | $8,364,000 |
| Annual $ThO_2$ Production | 1,528,000 |
| Unit Cost of Milling | $5.50/lb $ThO_2$ |

## TABLE 10.4. Wet Mountain Mill Capital Costs

| | |
|---|---|
| Total Depreciable Capital Investment | $9,572,000 |
| Land | 163,000 |
| Working Capital | 671,000 |
| TOTAL Capital Investment | $10,406,000 |

### 10.4.2.2 Mill Operating Costs

The components of mill operating costs are shown in Table 10.5. Costs for electric power and fuel oil are scaled down using the 0.7 scaling factor. Operating labor force is estimated at 59 men from data on uranium mills of similar size. Reagents and water consumption are scaled down linearly.

## TABLE 10.5. Wet Mountain Mill Operating Cost

Direct Costs

| | |
|---|---|
| Operating Labor (59 men) | $ 951,000 |
| Supervision | 143,000 |
| Maintenance and Repairs | 624,000 |
| Operating Supplies | 94,000 |
| Laboratory Charges | 143,000 |
| Reagents | 310,000 |
| Utilities | |
|     Fuel Oil | 234,000 |
|     Electricity | 169,000 |
|     Water | 27,000 |
| Sulfuric Acid Operating Plant | $ 779,000 |

Indirect Costs

| | |
|---|---|
| Plant Overhead | $1,031,000 |
| Administrative Costs | 258,000 |
| TOTAL Operating Costs | $4,763,000 |

## 10.5 MAJOR UNCERTAINTIES

Major areas of uncertainty affecting the $ThO_2$ unit cost estimates include mine capital and operating costs, mill capital and operating costs, mine capacity, mill capacity, and mill recovery efficiency. Our estimates for the range of uncertainty in these areas are:

| | |
|---|---|
| Mine Capital Cost | +100% to -10% of calculated value |
| Mine Operating Cost | +100% to -10% of calculated value |
| Mill Capital Cost | +50% to -10% of calculated value |
| Mill Operating Cost | +50% to -10% of calculated value |
| Mine Capacity | 2000 tpd to 200 tpd with most likely value of 350 tpd |
| Mill Capacity | 2000 tpd to 200 tpd with most likely value of 350 tpd |
| Recovery Efficiency | 96% to 40% with most likely value of 91% |

The impacts of these uncertainty ranges on the unit cost of $ThO_2$ is shown in Table 10.6. From the incremental unit cost values, a Monte Carlo Statistical Analysis is performed using the procedure described in Section 16. From this analysis, a most likely value of \$8.30/lb $ThO_2$ and overall cost range of \$5.92/lb to \$13.30/lb is predicted for the unit cost of $ThO_2$ at Wet Mountain.

TABLE 10.6. Uncertainty Ranges for Factors Affecting the $Tho_2$ Unit Cost for the Wet Mountains

| | Most Likely Value | Range |
|---|---|---|
| Mine Capital Cost $10^3$/year | \$1,232[a] | 2,654 to 1,153 |
| Unit Mine Capital Cost $/lb $ThO_2$ | .84 | 1.68 to .75 |
| Incremental Unit Cost $/lb $ThO_2$ | | .84 to -.09 |
| Mine Operating Cost $10^3$/year | 1,513 | 3,026 to 1,362 |
| Unit Mine Operating Cost $/lb $ThO_2$ | .99 | 1.98 to .89 |
| Incremental Unit Cost $/lb $ThO_2$ | | +.99 to -.10 |
| Mill Capital Cost $10^3$/year | 3,606 | 5,409 to 3,245 |
| Unit Mill Capital Cost $/lb $ThO_2$ | 2.36 | 3.54 to 2.12 |
| Incremental Unit Cost $/lb $ThO_2$ | | +1.18 to -.24 |
| Mill Operating Cost $10^3$/year | 4,763 | 7,144 to 4,287 |
| Unit Mill Capital Cost $/lb $ThO_2$ | 3.12 | 1.68 to 2.81 |
| Incremental Unit $/lb $Th0_2$ | | +1.56 to -.31 |
| Mine Capacity tpd | 350 tpd | 2,000 to 200 |
| Unit Mine Capacity Cost $/lb $ThO_2$ | | 7.01 to 6.68 |
| Incremental Unit Cost $/lb $ThO_2$ | | +.21 to -.12 |
| Mill Capacity tpd | 350 tpd | 2,000 to 200 |
| Unit Mill Capacity Cost $/lb $ThO_2$ | | |
| Incremental Unit Cost $/lb $ThO_2$ | | .89 to -2.03 |
| Mill Recovery Efficiency % | 91 | 40 to 96 |
| Unit Recovery Efficiency Cost $/lb $ThO_2$ | 7.31 | 16.63 to 6.93 |
| Incremental Unit Cost $/lb $ThO_2$ | | +9.32 to -.38 |

(a) Less 22% depletion allowance

# REFERENCES

1. M. H. Staatz, "Thorium Veins in the United States", Economic Geology, Vol. 69, pp. 494-507, 1974.

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3. F. J. Kelly, Technical and Economic Problems of Rare-Earth-Metal and Thorium Resources in Colorado, New Mexico, and Wyoming, USBM IC 8124, p. 38, 1961.

4. M. H. Staatz, United States Mineral Resources, USGS Prof. Paper #820, p. 474, 1973.

5. Chemical and Operating Cost Estimating System Handbook, Prepared for the United States Dept. of Interior, Bureau of Mines, by STRAAM Engineers, Inc., Arcadia, CA, 374 pp., July 1978.

## 11.0 THORIUM PRODUCTION FROM PALMER, MICHIGAN

### 11.1 RESOURCE DESCRIPTION

The Palmer area is located in the east-central portion of Marquette County on the northern peninsula of Michigan. The deposit occupies approximately 3 square miles and includes the town of Palmer. The Palmer area is characterized by relatively rugged, knobby topography covered by a dense forest. Thorium-bearing Goodrich quartzite forms the bulk of the ore deposit. This deposit contains an estimated 193,000 tons of $ThO_2$.

The Goodrich quartzite of the Palmer area is composed of a locally developed basal conglomerate which consists almost entirely of boulders and pebbles of granite. Above the basil unit, alternating beds of coarse quartzite and pebble conglomerate exist. The monazite concentrations are associated with the pebble conglomerates.

Since surface mining appears to be the most feasible mining method, the monazite is assumed to be evenly distributed throughout the quartzite. The thorium content is estimated at 1.06 lb/ton (.06%) but ranges from .26 to 3.56 lb/ton[1], while the average monazite content is estimated at 14.7 lbs/ton. The monazite is assumed to be 8.2% $ThO_2$.

Other heavy minerals associated with the quartzite are mainly hematite, magnetite, ilmenite, and rutile. No estimates are available as to the amount of these minerals. The rare earth oxide content of the monazite is approximately 46%[2], and, the amount of mineable rare earth oxides contained in the monazite at Palmer is estimated at over 1,000,000 tons. (See Appendix F).

### 11.2 MINE DESCRIPTION

Currently, the Palmer, Michigan deposit is not being mined for thorium, although the Empire Iron Mine is located one mile northwest of the Palmer townsite. The conceptual Palmer, Michigan mine was developed based on information about the deposit. As the Goodrich formation is near the surface, the Palmer, Michigan mine will utilize open pit mining.

The conceptual mine consists of a pit designed to remove the upper central part of the ore body which contains 123,500 tons of $ThO_2$ in 205 million tons of ore. The pit also contains 55 million tons of waste rock and till. Ten million tpy of ore will be produced from the pit along with about 3 million tons of waste. The mine is designed to operate 3 shifts/day, five days/wk for 240 days/yr for 20 yr. Mine equipment and shops are sized for these parameters. Ore recovery is assumed to be approximately 97%.

## 11.2.1  Mining Process

The major process steps for mining are described in the following section.

### 11.2.1.1  Removing the Overburden

Approximately 30 feet of overlying till must be removed before mining. The till is scraped using bulldozers and the overburden is loaded into trucks using front-end loaders. The waste is then hauled to a dump adjacent to the pit. The ore is now ready to be mined.

### 11.2.1.2  Drilling and Blasting

Blast holes are drilled into the ore that constitutes the Palmer deposit using a downhole hammer. The blast holes are loaded with explosives and detonated.

### 11.2.1.3  Loading the Ore

The ore is loaded into 120-ton electric wheel trucks using 15 $yd^3$ shovels. Fifteen cubic yard front end loaders will be provided also to clean up around the shovels.

### 11.2.1.4  Hauling of Ore

The electric ore trucks transport the carbonatite to a concentrator located approximately one-half mile from the mine. The truck is routed through a scanner to determine ore grade prior to unloading.

### 11.2.1.5  Effluent Control

The only radioactive solid waste is waste rock from the dry mill and the glacial till overburden. Waste management consists simply of stockpiling the waste rock in a storage pile equipped with wind barriers in a dump site adjacent to the pit.

The main liquid effluent stream leaving the mine is the mine water. This water is pumped out of the mine at the mill site to a holding pond where it is allowed to settle. It is then recycled for use in the mill or used to wet down haul roads for dust control. Access water will be allowed to drain to natural surface water drainage systems.

The airborne effluents are made up of dust and combustion gases (including blasting gas and equipment exhaust gas). These airborne effluents are given off during blasting, loading, blending, and crushing. The roads and ore piles are periodically wet down to minimize the amount of windblown dust.

## 11.3  MILL DESCRIPTION

Monazite conglomerate has not been processed for any commercial purpose, but laboratory studies have been conducted on the benefication of conglomerate monazite. The overall recoverability of monazite in the Goodrich Quartzite has been reported to be 85%[2], but no process description was available. Successful laboratory separation of monazite from the Deadwood Conglomerate found at Bald Mountain, Wyoming has also been reported with an overall recovery of 61%. We assume that the concentration process for Bald Mountain also applies to Goodrich Quartzite, thus Palmer, Michigan conglomerate milling utilizes the process described by Borrowman and Rosenbaum[3].

The Palmer, Michigan mill processes 41,700 tpd of ore at the primary crusher and produces approximately 290 tpd of concentrate. The concentrate contains 5.2% $ThO_2$. The process recovers 61% of the thorium originally in the ore.

### 11.3.1  Milling Process

The conglomerate milling process consists of crushing followed by gravity and magnetic separation. The process steps for the reference conglomerate mill are described in the following sections. A detailed process flow diagram is shown in Figure 11.1.

11.3

FIGURE 11.1. Palmer Michigan Mill

11.4

# FLOW STREAMS FOR PALMER, MICHIGAN ORE MILL

1. 1895.45 tons/hr ore w/$^{232}$Th = .98912 ton/hr
2. 189.545 lb/hr w/$^{232}$Th = .0989 lb/hr
3. 1377.27 tons/hr ore w/$^{232}$Th = .7187 ton/hr
4. 137.7 lb/hr w/$^{232}$Th = .073 lb/hr
5. 1377.20 tons/hr w/$^{232}$Th = .7187 ton/hr
6. 8656.96 tons/hr $H_2O$ and 1347.04 tons/hr ore w/$^{232}$Th = .2762 ton/hr
7. 311.13 tons/hr $H_2O$ and 1225.78 tons/hr ore w/$^{232}$Th = .2488 ton/hr
8. 779.12 tons/hr $H_2O$ and 121.26 tons/hr ore w/$^{232}$Th = .0274 ton/hr
9. 1092.65 tons/hr $H_2O$
10. 8659.36 tons/hr $H_2O$
11. 2.40 tons/hr $H_2O$ and 30.16 tons/hr ore w/$^{232}$Th = .4425 ton/hr
12. 2.40 tons/hr $H_2O$ and 109.98 lb/hr ore w/$^{232}$Th = .057 lb/hr
13. 20.60 tons/hr w/$^{232}$Th = 8.40 lb/hr
14. 9.505 tons/hr w/$^{232}$Th = 876.5 lb/hr
15. 0.95 lb/hr w/$^{232}$Th = .044 lb/hr
16. 9.505 tons/hr w/$^{232}$Th = 876.5 lb/hr

FIGURE 11.1. (cont'd)

### 11.3.1.1 Run-of-Mine Storage

Ore arrives at the mill by truck and is dumped in a storage area. The ore is segregated according to grade, blended to maintain a relatively constant input to the mill, and then transferred from the storage area to the primary crusher conveyor by means of a front-end loader.

### 11.3.1.2 Crushing and Grinding

The ore is initially crushed in a primary crusher and then transferred to the secondary crusher, which grinds the ore to minus 20 mesh. The discharge is screened and the oversized particles are returned to the secondary crusher. The processed ore is stored in a crushed-ore bunker.

### 11.3.1.3 Gravity Separation

The crushed ore is mixed with water and is pumped to a rougher shaking-table circuit. The tails are discarded to the tails pond and the milling

product is processed in the scavenger shaking table circuit. The scavenger shaking table concentrate is returned to the rougher circuit for reprocessing. The rougher circuit concentrate is transferred to the wet-concentrate storage pile for dewatering.

### 11.3.1.4 Drying

The wet concentrate is transferred from the wet concentrate storage to a rotary dryer using an overhead crane. The concentrate is dryed and discharged to the magnetic separator belt conveyor.

### 11.3.1.5 Magnetic Separation

The dry concentrate is processed by rougher cross-belt magnetic separators. The tails from the rougher magnetic separator are processed by smaller scavengers magnetic separators in parallel. The tails are discarded and while the concentrate is recycled to the rougher magnetic separator. The final concentrate obtained is discharged to storage until it is shipped to the refinery.

## 11.3.2 Effluent Control

Two types of effluents are generated from the mill. Airborne wastes generated during unloading, crushing and grinding are controlled using 950,000 cyclone dust collection systems. Wet impingement scrubbers control dust from loading, drying, and magnetic separation.

Liquid effluents generated during gravity and magnetic separation are piped to a 200 acre settling pond. Water from the tails pond is recycled to the mill for use in gravity separators.


## 11.4 REFINERY DESCRIPTION

While placer monazite has been successfully mined and refined, conglomerate monazite has not. The conceptual refinery is, therefore, based on a laboratory scale process developed by the United States Bureau of Mines[3]. The reference mill has a design capacity of 300 tons of concentrate per day. The process recovers 95% of the $ThO_2$ and produces a product of 99+% purity. The refinery operates 24 hours/day for 330 days/year for 20 years. The

refinery is assumed to be located approximately 10 miles from the Palmer mine/mill site.

## 11.4.1  Refining Process

Acid leaching, followed by solvent extraction and caustic digestion, constitute the refining process.  The following sections discuss the major steps involved in the refining of conglomerate monazite.  A detailed process flow diagram is shown in Figure 11.2.

### 11.4.1.1  Grinding

The concentrate arrives from the mill by truck and is stored until processing.  The concentrate, which has already been crushed to 20-mesh, contains between 1% and 6% thorite.  Before leaching, the concentrate is dry ground to 100-mesh in a ball mill.

### 11.4.1.2  Leaching

The concentrate is mixed with 1.5 times its weight of 96% $H_2SO_4$ and heated to 200°C in agitated tanks for 2 hr.  The concentrate is then mixed with 5 parts water and agitated for 1 hr.  The pregnant solution is filtered and washed with acidified water.

### 11.4.1.3  Liquid/Solid Separation

After the acid leaching process the pregnant solution contains 6% solids. It is flocculated with 0.2 lb of flocculate per ton of concentrate.  After filtration using a leaf type pressure filter, the pregnant liquor is then transferred to the solvent extract process.

### 11.4.1.4  Solvent Extraction

Thorium in the leach liquor is extracted using an organic solvent with 10% aliphatic primary amine, 5% Primary decyl alcohol (PDA) and 85% kerosene. The leach liquor and the organic solvent are brought into contact in the mixer settler for 15 minutes.  The barren leach solution is discarded and the loaded organic is scrubbed in mixer/settlers with 0.2 M sulfuric acid for removal of rare earth oxides.

CONGLOMERATE REFINERY

FIGURE 11.2. Palmer Michigan Refinery

11.8

1.  9.505 tons/hr concentrate w/$^{232}$Th = 876.48 lb/hr
2.  .9505 lb/hr w/$^{232}$Th = .044 lb/hr
3.  9.505 tons/hr w/$^{232}$Th = 876.436 lb/hr
4.  16.01 tons/hr $H_2SO_4$
5.  25.51 tons/hr w/$^{232}$Th = 876.436 lb/hr
6.  47.525 tons/hr $H_2O$
7.  73.035 tons/hr w/$^{232}$Th = 876.436 lb/hr
8.  1.90 lb/hr flocculant
9.  24.71 tons/hr $H_2O$
10.  48.47 tons/hr wash solution w/$^{232}$Th = 0.7788 lb/hr
11.  26.46 tons/hr w/$^{232}$Th = 30.254 lb/hr
12.  121.505 tons/hr w/$^{232}$Th = 846.96 lb/hr
13.  71.29 tons/hr organic (10% JM-T, 5% PDA, 85% kerosene)
14.  121.505 tons/hr w/$^{232}$Th = 1.55 lb/hr
15.  71.29 tons/hr w/$^{232}$Th = 845.41 lb/hr
16.  23.76 tons/hr of scrub solution (0.2M $H_2SO_4$)
17.  23.76 tons/hr w/$^{232}$Th = 0.779 lb/hr
18.  71.29 tons/hr w/$^{232}$Th = 844.63 lb/hr
19.  38.02 tons/hr strip solution (1.5M NaCl, .5M HCl)
20.  38.02 tons/hr w/$^{232}$Th = 844.63 lb/hr
21.  .9505 ton/hr $Na_2CO_3$
22.  38.9705 tons/hr w/$^{232}$Th = 844.63 lb/hr
23.  38.02 tons/hr w/$^{232}$Th = 0.779 lb/hr
24.  .9505 ton/hr w/$^{232}$Th = 843.85 lb/hr
25.  285.15 lb/hr NaOH
26.  1.90 tons/hr $H_2O$
27.  2.993 tons/hr w/$^{232}$Th = 843.85 lb/hr
28.  1.90 tons/hr wash solution
29.  3.80 tons/hr w/$^{232}$Th = .080 lb/hr
30.  1.093 tons/hr w/$^{232}$Th = 843.77 lb/hr
31.  0.609 tons/hr w/$^{232}$Th = 0.803 lb/hr
32.  968.0 lb/hr w/$^{232}$Th = 842.967 lb/hr
33.  0.1596 lb/hr w/$^{232}$Th = .139 lb/hr
34.  967.84 lb/hr w/$^{232}$Th = 842.831 lb/hr
35.  5.7 tons/hr $NaSO_4$
36.  127.205 tons/hr w/$^{232}$Th = 1.55 lb/hr
37.  94.9 tons/hr w/$^{232}$Th = 1.05 lb/hr
38.  32.3 tons/hr w/$^{232}$Th = 0.50 lb/hr
39.  9.1 tons/hr NaOH (30%)
40.  41.4 tons/hr w/$^{232}$Th = .50 lb/hr
41.  10.2 tons/hr $H_2O$
42.  41.0 tons/hr w/$^{232}$Th = .11 lb/hr
43.  10.6 tons/hr w/$^{232}$Th = 0.39 lb/hr
44.  3.2 tons/hr w/$^{232}$Th = .0062 lb/hr
45.  7.4 tons/hr w/$^{232}$Th = .384 lb/hr (46% REO)

<u>FIGURE 11.2.</u>  (cont'd)

11.9

### 11.4.1.5 Salt Stripping

A salt solution is used to strip the thorium from the loaded organic. The loaded organic and the salt solution are brought into contact in three stainless steel agitated tanks. The premixed salt solution containing 1.5 normal sodium chloride and 0.5 normal hydrochloride acid removes the thorium from the organic. The organic is returned to the amine extraction step. The loaded strip liquor is pumped to the neutralization process.

### 11.4.1.6 Neutralization

The loaded strip liquor is neutralized with soda ash in an agitated tank and the thorium precipitates out of solution. The precipitate is dewatered using a rotary drum filter. The filtrate is discarded.

### 11.4.1.7 Caustic Digestion

The dewatered precipitate is pumped to a stainless steel agitated tank where the precipitate is digested with caustic soda at room temperature. The caustic digestion forms hydrus thorium oxide which is removed from the caustic solution with a continuous rotary drum filter. The filtrate is discarded.

### 11.4.1.8 Drying and Ignition

The hydrous thorium oxide is dried to 10% moisture content in a twin-screw dryer which operates with a jacket temperature of 120°C and a screw temperature of 154°C. The twin-screw dryer discharges thorium hydroxide into a rotary kiln where it is converted to $ThO_2$. The kiln operates at 850°C and runs 7% full[4]. The discharge from the rotary kiln is $ThO_2$ of 99+% purity. The $ThO_2$ is packaged in drums and shipped to the fabrication facility.

### 11.4.1.9 Byproduct Recovery

The raffinate from the solvent extraction phase contains the rare earth oxides. To precipitate the rare earths, 60 g/ℓ of sodium sulfate is added to the raffinate. Ninety-eight percent of the rare earth byproducts are converted to the sulfate salt. The rare earth sulfates are treated with caustic soda, and a rare earth hydroxide precipitate is formed. The precipitate is dried at 140°C. Product purity is estimated at 46% rare earth oxides[3].

## 11.4.2 Effluent Control

Effluent control methods are similar to those utilized for the Lemhi Pass refinery. They are described in Section 8.3.2 of this report.

## 11.4.3 Sulfuric Acid Plant

In order to reduce the cost of sulfuric acid, an on-site sulfuric acid plant is included with the refinery.

## 11.5 PALMER, MICHIGAN PRODUCTION COSTS

The production costs for developing an estimated 70,000 tons of $ThO_2$ from Palmer, Michigan is $12.88/lb $ThO_2$. After taking into account uncertainties in mine capital and operating costs, refinery capital and operating costs, and thorium recovery, a unit cost of $14.20/lb $ThO_2$ is estimated from the method described in Section 16. These costs, (Table 11.1) are developed in the following sections.

TABLE 11.1. Palmer Production Cost Summary

|  | Unit Cost ($/lb $ThO_2$) |
|---|---|
| Mining/Milling | 10.90 |
| Refining | 2.00 |
| Total from Estimates | 12.90 |
| Total Including Uncertainty | 14.20 |

## 11.5.1 Mine/Mill Production Costs

The production costs for Palmer, Michigan mines are shown in Table 11.2. Capital and operating costs were developed by a mining consulting firm and are shown in Appendix E. A fixed charge rate of .2338 was assumed to amortize capital investment over 20 years.

TABLE 11.2. Palmer Mine/Mill Production Costs

Annual Fixed Charges:

| | |
|---|---|
| Depreciable Capital | $42,792,000 |
| Non-Depreciable Capital | 2,123,000 |
| Annual Operating Costs | 44,747,000 |
| Subtotal | $89,662,000 |
| Depletion Allowance | 19,726,000 |
| Annual Production Costs | $69,936,000 |
| Annual Levelized Production of $ThO_2$ | 6,432,000 lbs |
| Unit Cost of Mining/Milling | $10.87/lb |

### 11.5.1.1 Mine/Mill Capital Costs

Capital costs for the mine and mill were determined jointly. (See Appendix E). Total capital costs are approximately $204,258,000 (Table 11.3).

TABLE 11.3. Palmer Mine/Mill Capital Cost

| | |
|---|---|
| Mine/Mill Buildings and Equipment | $159,720,000 |
| Effluent Control Buildings and Equipment Cost | 407,000 |
| Mine Tailings Pond | 236,000 |
| Mill Tailings Pond | 7,220,000 |
| Exploration | 2,977,000 |
| Development | 11,462,000 |
| Environmental Impact | 1,006,000 |
| Total Depreciable Capital Investment | $183,028,000 |
| Land | 21,230,000 |
| Total Capital Investment | $204,258,000 |

### 11.5.1.2 Palmer Mine/Mill Operating Costs

Combined operating costs for the mine and mill are approximately $44,747,000 annually (Table 11.4). These costs were developed in Appendix E.

TABLE 11.4. Palmer Mine/Mill Operating Costs

| | |
|---|---|
| Mine Labor Costs[a] | $ 6,006,000 |
| Materials and Supplies | 7,770,000 |
| Mine Equipment Replacement | 2,046,000 |
| Mill Labor Costs[a] | 6,424,000 |
| Materials and Supplies | 12,848,000 |
| Mill Equipment Replacement | 7,000,000 |
| Mine/Mill Effluent Control Annual Cost | 2,653,000 |
| TOTAL | $44,747,000 |

(a) Includes overhead.

### 11.5.1.3 Refinery Production Costs

The refining costs for the Palmer, Michigan ore is $2.00/lb $ThO_2$. (Table 11.5) A fixed charge rate of .2338 was applied to nondepreciable capital investment. A fixed charge of 10% was applied to land and working capital.

TABLE 11.5. Palmer Refinery Production Costs

| | |
|---|---|
| Annual Fixed Charges: | |
| Depreciable Capital | $ 3,935,000 |
| Non-Depreciable Capital | 217,000 |
| Annual Operating Cost | 8,519,000 |
| Subtotal | $12,671,000 |
| Annual Transportation Cost | 320,000 |
| Annual Production Cost | $12,991,000 |
| Annual Levelized Production of $ThO_2$ | 6,432,000 lbs |
| Unit Cost of Refining | $2.00/lb $ThO_2$ |

### 11.5.1.4 Refinery Capital Cost

The total capital investment for the Palmer, Michigan refinery is estimated to be $19,000,000 (Table 11.6). The refinery capital cost and effluent

control equipment capital cost were estimated using the procedure described in Section 6.0. The capital cost of the sulfuric acid plant was estimated from Guthrie[6] based on annual acid consumption of 110,000 tpd.

TABLE 11.6. Palmer Refinery Capital Costs

| | |
|---|---:|
| Refinery Buildings and Equipment Cost | $ 9,116,000 |
| Effluent Control Buildings and Equipment Cost | 146,000 |
| Sulfuric Acid Plant Buildings and Equipment Cost | 4,123,000 |
| Access Road | 100,000 |
| Feasibility Study | 100,000 |
| Tailings Pond | 1,216,000 |
| Environmental Impact | 100,000 |
| Contingency (15% of DPC) | 1,941,000 |
| TOTAL Depreciable Capital Investment | $16,832,000 |
| Land | 227,000 |
| Working Capital (15% of DPC) | 1,941,000 |
| TOTAL Capital Investment | $19,000,000 |

### 11.5.1.5 Refinery Operating Costs

The annual operating costs for the 300 tpd Palmer, Michigan, conglomerate refinery are estimated to be $8,519,000 (Table 11.7). Maintenance supplies, plant supplies, and overheads were estimated using the procedure described in Section 6.0. Refinery labor, reagent expenses, utilities, and sulfuric acid plant operating costs will be discussed below.

### 11.5.1.6 Refinery Labor Cost

The refinery operating labor force was derived from the operator requirements for each piece of equipment. The total operating labor was calculated by adding the labor requirements of each component. Individual component requirement was estimated from Blecker and Nichols[5]. The total requirement for operators was estimated to be 11 men/shift which includes two floaters. Hourly cost of operating labor is assumed to be $8.06/hr.

TABLE 11.7.  Palmer Refinery Operating Costs

Direct Costs

| | |
|---|---:|
| Operating Labor (33 men) | $   702,000 |
| Supervisory and Clerical | 105,000 |
| Maintenance and Repairs | 1,140,000 |
| Operating Supplies | 171,000 |
| Laboratory Charges | 105,000 |

Utilities

| | |
|---|---:|
| Electricity | 306,000 |
| Fuel Oil | 473,000 |
| Water | 21,000 |
| Reagents | $ 2,494,000 |

Indirect Costs

| | |
|---|---:|
| Plant Overhead | $ 1,168,000 |
| Administrative Overhead | 292,000 |
| Sulfuric Acid Operating Cost | $ 1,542,000 |
| TOTAL Operating Cost | $ 8,519,000 |

## 11.5.1.7  Reagent Expense

Reagent expenses were calculated by estimating the annual consumption of each reagent and multiplying by the current cost of the reagent.  Since a sulfuric acid plant is provided, no expense for sulfuric acid is included. Table 11.8 presents reagent consumption, unit cost and annual reagent cost for the monazite conglomerate refinery.  The total reagent expense was estimated to be $2,494,000.

## 11.5.1.8  Sulfuric Acid Plant Operating Expense

The operating expenses for the sulfuric acid plant (raw materials, operating supplies, maintenance materials, labor, supervision, utilities, and indirect costs) were estimated from Guthrie[6] for an 110,000 ton/day sulfuric acid plant.  The operating cost was estimated to be $1,542,000.

TABLE 11.8.  Palmer Reagent Consumption and Expense

| Reagent | Annual Consumption | Annual Consumption | Unit Cost | Annual Cost |
|---------|--------------------|--------------------|-----------|-------------|
| NaCl | 6,700 lb/hr | 26,500 tons/yr | $ 41/ton | $ 1,086,000 |
| HCl | 1,400 lb/hr | 5,500 tons/yr | $ 61/ton | $ 335,000 |
| Soda Ash | 1,900 lb/hr | 7,524 tons/yr | $ 69/ton | $ 519,000 |
| NaOH | 290 lb/hr | 1,100 tons/yr | $480/ton | $ 528,000 |
| Kerosene | 8.1 gals/hr | 64,152 gal/yr | $0.4/gal | $ 26,000 |
| | TOTAL Annual Reagent Expense | | | $ 2,494,000 |

## 11.5.1.9  Utilities

Electric power demand was calculated by adding the electric power demand for each equipment component.  The annual power cost was calculated to be $306,000 assuming a power factor of 0.9, a distribution factor of 0.9, and a power cost of $0.03/kWh.

No. 6 fuel oil demand was estimated by calculating the fuel oil demand for the utility boiler and rotary calciner.  The utility boiler was sized to provide steam for space heating, the twin screw dryer, and heating of the leaching tanks.  The total demand for fuel oil was 946,000 gpy with an assumed cost of $0.5/gal.  The annual cost of No. 6 fuel oil is $473,000.

Water demand was calculated from the process flow charts as 140,000,000 assuming a cost of $0.15/1000 gallons[4].  The annual expense for water is $21,000.

## 11.5.1.10  Transportation Cost

The transportation cost for hauling the ore from the mill to the refinery is assumed to be $300,000 per year.  Assuming a haul distance of 10 miles, two 30 ton haul trucks will move 290 tpd of concentrate.  The capital cost of the trucks is estimated to be $320,000.  The useful life of the trucks is assumed to be 5 years.  The operating costs are summarized in Table 11.9.

TABLE 11.9. Transportation Cost

Annual Transportation Cost

| | |
|---|---|
| Fixed Charge | $110,000 |
| Operating Labor | 120,000 |
| Labor Overhead | 40,000 |
| Maintenance | 20,000 |
| Fuel | 30,000 |
| TOTAL | $320,000 |

## 11.5.2 Byproduct Credit

Due to the high consumption of sodium hydroxide necessary to precipitate the rare earths, they are not economic to recover. Operating costs would exceed revenue by approximately $5,100,000/yr or $.80/lb $ThO_2$. The conceptual process to recover rare earths is discussed in the following sections.

### 11.5.2.1 Byproduct Production Cost

The incremental production cost of byproducts is approximately $16,130,000 (Table 11.10). A fixed charge rate of .2338 was applied to depreciate capital investment over 20 years. A fixed charge of 10% was applied to land and working capital.

TABLE 11.10. Incremental Cost of Byproducts

| | |
|---|---|
| Annual Fixed Charges: | |
| Depreciable Capital | $ 1,321,000 |
| Non-Depreciable Capital | 79,000 |
| Annual Operating Cost | 14,730,000 |
| Annual Production Cost | $16,130,000 |

### 11.5.2.2 Capital Costs for Rare Earth Recovery

The estimated additional capital cost associated with recovering byproducts is summarized in Table 11.11. This cost is approximately $6,437,000.

TABLE 11.11. Components of Byproduct Capital Cost

| | |
|---|---|
| Buildings and Equipment Cost | $4,847,000 |
| Feasibility Study | 100,000 |
| Contingency | 704,000 |
| Total Depreciable Capital Investment | $5,651,000 |
| Land | 82,000 |
| Working Capital | 704,000 |
| TOTAL Capital Investment | $6,437,000 |

11.5.2.3 Operating Cost for Rare Earth Recovery

Incremental byproduct operating costs are approximately $14,730,000 (Table 11.12). Labor costs are based on the number of operators necessary for equipment operation. Supervision, maintenance, repairs, operating supplies, laboratory charges, utilities, and indirect costs are estimated using the procedure described in Section 6.0. Reagent expense is discussed below.

TABLE 11.12. Annual Operating Costs of Byproduct Recovery

| | |
|---|---|
| Direct Costs | |
| Operating Labor (4 man/shift) | $ 271,000 |
| Supervision | 41,000 |
| Maintenance and Repairs | 386,000 |
| Operating Supplies | 57,000 |
| Utilities | 500,000 |
| Laboratory Charges | 41,000 |
| Reagents | 12,910,000 |
| | $14,206,000 |
| | |
| Indirect Costs | |
| Plant Overhead | 419,000 |
| Administrative Overhead | 105,000 |
| | $14,730,000 |

### 11.5.2.4  Reagent Expense

Reagent cost estimates are based on process information[3] and theoretical consumption requirements.  These costs are summarized below.

| Reagent | Consumption | Unit Cost | Annual Cost |
|---------|-------------|-----------|-------------|
| NaOH | 22,000 tons/yr | $480/ton | $10,560,000 |
| $Na_2SO_4$ | 45,000 tons/yr | $51.9/ton | 2,350,000 |
| | | | $12,910,000 |

### 11.6 MAJOR UNCERTAINTIES

Major areas of uncertainty affecting the $ThO_2$ unit cost estimates include mine/mill capital and operating costs, refinery capital and operating costs, and thorium recovery.  Our estimates for the ranges of uncertainty in these areas are:

| | |
|---|---|
| Mine/Mill Capital Cost | +50% to -10% of calculated value |
| Mine/Mill Operating Cost | +50% to -10% of calculated value |
| Refinery Capital Cost | +50% to -10% of calculated value |
| Refinery Operating Cost | +50% to -10% of calculated value |
| Thorium Recovery Efficiency | 50% to 62% with a most likely value of 56% |

The impact of these uncertainty ranges on the unit cost of $ThO_2$ is shown in Table 11.13.  From the incremental unit cost values, a Monte Carlo Statistical Analysis is performed using the procedure described in Section 16.  From this analysis, a most likely value of $14.20/lb $ThO_2$ and overall cost range of $9.90/lb to 17.10/lb $ThO_2$ is predicted for the unit cost of $ThO_2$.

TABLE 11.13.  Uncertainty Ranges for Factors Affecting $ThO_2$ Unit Cost

| | Most Likely Value | Range |
|---|---|---|
| Mine/Mill Capital Cost $10^3$/year | 35,034[a] | 52,551 to 31,530 |
| Unit Mine/Mill Capital Cost $/lb $ThO_2$ | 5.45 | 8.17 to 4.90 |
| Incremental Unit Cost $/lb $ThO_2$ | | +2.72 to -.55 |
| Mine/Mill Operating Cost $10^3$/year | 34,903[a] | 52,354 to 31,412 |
| Unit Mine/Mill Operating Cost $/lb $ThO_2$ | 5.43 | 8.14 to 4.88 |
| Incremental Unit Cost $/lb $ThO_2$ | | +2.71 to -.55 |
| Refinery Capital Cost $10^3$/year | 4,152 | 6,228 to 3,737 |
| Unit Refinery Capital Cost $/lb $ThO_2$ | .65 | .97 to .58 |
| Incremental Unit Cost $/lb $ThO_2$ | | +.32 to -.07 |
| Refinery Operating Cost $10^3$/year | 8,519 | 12,779 to 7,667 |
| Unit Refinery Operating Cost $/lb $ThO_2$ | 1.32 | 1.99 to 1.19 |
| Incremental Unit Cost $/lb $ThO_2$ | | +.67 to -.13 |
| Thorium Recovery Efficiency (%) | 56 | 50 to 62 |
| Unit Recovery Efficiency Cost $/lb $ThO_2$ | 12.88 | 14.43 to 11.63 |
| Incremental Unit Cost $/lb $ThO_2$ | | +1.55 to -1.25 |

_____

(a) Depletion allowance (22%) is subtracted from cost

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# 12.0 THORIUM PRODUCTION FROM BEAR LODGE, WYOMING

## 12.1 RESOURCE DESCRIPTION

Located about 6 miles north of Sundance, Wyoming and occupying 1.5 square miles, the Bear Lodge deposit contains about 228,000 tons of $ThO_2$. Disseminated veinlets of thorium occur in iron-manganese zones. The grade of rock containing the veinlets ranges from 46 ppm $ThO_2$ to 1,200 ppm $ThO_2$. Principal identified thorium-bearing minerals are brockite, monazite and thorite. Rare earths are also contained in the mineralization at concentrations ranging from 12,000 to 18,000 ppm total rare earth oxides.[1]

## 12.2 MINE DESCRIPTION

Three possible sites for an open pit mine are located in the north, central, and south blocks of the Bear Lodge deposits. The average grade of the north block is .023% $ThO_2$ and .75% rare earth oxides. The central block averages .042% $ThO_2$ and 1.71% rare earth oxides. The south block contains .035% $ThO_2$ and 1.35% rare earth oxides[1].

The open pit mines shall be modeled after the Palmer, Michigan mine. Each mine shall operate 3 shifts/day, 5 days/week, for 240 days/year. Since tonnages for each pit are unknown, we assume approximately 51,700 tpd of material will be mined from each pit to recover 41,700 tpd. Thus, a total of 155,000 tpd of material or 125,000 tpd of ore will be mined.

### 12.2.1 Mining Process

The mining process for the Bear Lodge mine is similar to the process utilized for Palmer, Michigan mine. The mining process is described in Section 11.2.1 of this report.

## 12.3 MILL DESCRIPTION

The Bear Lodge deposit is not currently processed for thorium and laboratory studies on the benefication of thorium are not available. Since the mineralization at Bear Lodge is extremely fine grained, many separations processes such as gravity and flotation would be impractical. Thorium is,

therefore, assumed to be submitted to the milling process developed for the thorites using a dilute, sulfuric acid leach. The thorite would probably be the primary mineral amenable to acid attack. Recovery is assumed to be 60% because the thorium contained in the monazite would be almost inert to the low concentration of acid.

The Bear Lodge mill is assumed to process 125,000 tpd of ore and produce 25 tpd of $ThO_2$. The mill operates 3 shifts/day, 240 days/yr, for 20 yr and produces $ThO_2$ of 99+% purity.

## 12.3.1 Milling Process

The milling process is described in Section 8.3.1 of this report. However we assumed the equipment would be modified to account for the large scale of this operation. Therefore, equipment will be comparable to the Conway Granite refining equipment described in Section 14.3.1 of this report.

## 12.4 BEAR LODGE PRODUCTION COST

The production costs for developing an estimated 137,000 tons of $ThO_2$ from Bear Lodge range from $53.20/lb to $56.80/lb. After taking into account uncertainties in mine capital and operating costs, mill capital and operating costs, and thorium recovery efficiency, a most likely cost of $56.70 to 57.80 was determined by the method described in Section 16. The costs from Table 12.1 are developed in the following sections.

TABLE 12.1. Bear Lodge Production Cost Summary

|  | North | Central | South |
|---|---|---|---|
| Mining | $ 7.86 | $ 4.31 | $ 5.17 |
| Milling | 48.92 | 48.92 | 48.92 |
| Total from Estimates | $56.80 | $ 53.20 | $54.10 |
| Total (Including Uncertainty) | $57.80 | $ 56.80 | $57.70 |

## 12.4.1 Mine Production Cost

The production cost for mining the Bear Lodge resource range from $4.30/lb $ThO_2$ to $7.90/lb $ThO_2$ (Table 12.2). A fixed charge rate of .2338 is applied to annualize the capital investments over 20 yr.

12.2

# TABLE 12.2. Bear Lodge Mine Production Costs

## North Block

| | |
|---|---|
| Annual Fixed Charges on Capital Investment | |
| Depreciable Assets | $ 8,127,000 |
| Non-Depreciable Assets | 900,000 |
| Annual Operating Costs | 15,822,000 |
| Annual Production Cost | $24,849,000 |
| Depletion | 5,467,000 |
| Total Annual Production Cost | $19,382,000 |
| Annual Levelized Production of $ThO_2$ | 2,466,000 |
| Unit Cost of Mining | $7.90/lb $ThO_2$ |

## Central Block

| | |
|---|---|
| Annual Fixed Charge on Capital Investment | |
| Depreciable Assets | $ 8,127,000 |
| Non-Depreciable Assets | 900,000 |
| Annual Operating Cost | 15,822,000 |
| Annual Production Cost | $24,849,000 |
| Depletion | 5,467,000 |
| Total Production Cost | $19,382,000 |
| Annual Levelized Production of $ThO_2$ | 4,502,000 |
| Unit Cost of Mining | $4.30/lb $ThO_2$ |

## South Block

| | |
|---|---|
| Annual Fixed Charge on Capital Investment | |
| Depreciable Assets | $ 8,127,000 |
| Non-Depreciable Assets | 900,000 |
| Annual Operating Cost | 15,822,000 |
| Annual Production Cost | $24,849,000 |
| Depletion | 5,467,000 |
| Total Production Cost | $19,382,000 |
| Annual Levelized Production of $ThO_2$ | 3,752,000 |
| Unit Cost of Mining | $5.20/lb $ThO_2$ |

12.3

### 12.4.1.1 Mine Capital Costs

Mine capital costs are estimated from the Palmer, Michigan mine model to be approximately $43,761,000. Cost per pit is summarized in Table 12.3.

TABLE 12.3. Mine Capital Cost

| | |
|---|---|
| Mine Buildings and Equipment Cost | $ 5,720,000 |
| Mine Tailings Pond | 236,000 |
| Exploration | 2,827,000 |
| Development | 25,588,000 |
| Environmental Impact | 390,000 |
| Total Depreciable Capital Investment | $34,761,000 |
| Land | 9,000,000 |
| Total Capital Investment | $43,761,000 |

### 12.4.1.2 Mine Operating Cost

Operating costs for the mine are approximately $15,822,000 annually (Table 12.4). These costs are assumed from the Palmer, Michigan mine model.

TABLE 12.4. Bear Lodge Mine Operating Costs

| | |
|---|---|
| Mine Labor Costs | $ 6,006,000 |
| Material and Supplies | 7,770,000 |
| Mine Equipment Replacement | 2,046,000 |
| | $15,822,000 |

### 12.4.2 Mill Production Costs

The milling/refining costs for Bear Lodge are $48.90/lb $ThO_2$. A fixed charge of .2338 was applied to nondepreciable capital investment. A charge of 10% was applied to land and working capital (Table 12.5).

TABLE 12.5. Bear Lodge Mill Production Costs

| | |
|---|---|
| Annual Fixed Charge | $ 56,133,000 |
| Annual Operating Cost | 467,712,000 |
| Subtotal | $523,845,000 |
| Annual ThO$_2$ Production | 10,709,000 lb |
| Unit Cost | $48.90/lb ThO$_2$ |

## 12.4.2.1 Mill Capital Costs

Mill equipment cost is estimated from both the Conway Granite mill and by scaling up of the Lemhi Pass mill with a .70 scaling factor. Costs are shown below:

| | |
|---|---|
| Conway Granite | $156,000,000 |
| Lemhi Pass | $251,000,000 |

These costs only include mill equipment and facilities.

Since the Conway Granite costs do not include the calcining step, they are assumed to be low. The Lemhi Pass value is assumed to be high because of the substantial economies of scale that would be realized. Thus, an average value of $200,000,000 will be assumed. Additional charges include (1) $100/day/ton for tailings disposal, (2) an additional $1 million for environmental impact studies, and (3) a contingency fee. Mill capital cost is thus approximately $240,000,000.

## 12.4.2.2 Mill Operating Cost

Mill operating costs are estimated from Conway Granite operating costs and adjusted to reflect a capacity of 90,900 tons per day of ore. Costs are approximately $467,712,000 (Table 12.6).

### Mill Labor Cost

Mill labor cost is derived from data on the Conway Granite mill. Approximately 87 men[2] are required for acid leaching, solvent extraction and precipitation steps assuming a highly automated processing circuit. An estimated 13 men will also be employed to handle the calcination and packaging operation.

TABLE 12.6.  Bear Lodge Mill Operating Costs

| | |
|---|---:|
| Operating Labor (100 men) | $  2,130,000 |
| Direct Supervisory and Clerical | 320,000 |
| Maintenance and Repairs | 2,086,000 |
| Utilities | 15,024,000 |
| Reagents | 444,117,000 |
| Operating Supplies | 313,000 |
| Laboratory Charges | 320,000 |
| Plant Overhead | 2,722,000 |
| Administrative Overhead | 680,000 |
| Total | $467,712,000 |

## Reagents

The largest cost consideration in determining reagent expense is the sulfuric acid consumption.  Based on data from the U.S.B.M.[3], the process will utilize approximately 450 lb of acid per ton of ore.  Consumptions for all other reagents is determined from the thorite milling process[4].  Reagent costs are summarized on Table 12.7.

TABLE 12.7.  Reagent Consumption and Expense

| Reagent | Annual Consumption | Unit Cost | Annual Cost |
|---|---|---|---:|
| Sulfuric Acid | 6,750,000 | $45/ton | 303,750,000 |
| Lime | 4,185,000 | $32/ton | 133,920,000 |
| HCl | 15,000 tons | $61/ton | 915,000 |
| Soda Ash | 18,000 tons | $69/ton | 1,242,000 |
| NaCl | 60,000 tons | $41/ton | 2,460,000 |
| $NaSO_4$ | 7,500 tons | $52/ton | 390,000 |
| $N_2OH$ | 3,000 tons | $.24/lb | 1,440,000 |
| | | | 444,117,000 |

### Utilities

Utilities consumption is based on data from the Conway Granite refinery. Approximately 10.7 kWh/ton ore will be required at a total cost of $9,630,000 annually[2]. Water consumption is approximately 5 tons/ton ore[4] from the thorite milling process. Cost for water is approximately $5,394,000 annually.

#### 12.4.2.3 Byproduct Credit

Since the major rare earth bearing mineral, monazite, is almost inert to dilute sulfuric acid leach, most of the rare earths cannot be recovered using the process outlined in this section.

### 12.5 MAJOR UNCERTAINTIES

Major areas of uncertainty affecting the $ThO_2$ unit cost estimates include mine capital and operating costs, mill capital and operating costs, and thorium recovery efficiency. Our estimate for the ranges of uncertainties in these areas are:

| | |
|---|---|
| Mine Capital Cost | +50% to -10% of calculated value |
| Mine Operating Cost | +50% to -10% of calculated value |
| Mill Capital Cost | +50% to -10% of calculated value |
| Mill Operating Cost | +60% to - 45% of calculated value |
| Thorium Recovery Efficiency | 40% to 85% with a most likely value of 60% |

The impact of there uncertainty ranges on the unit cost of $ThO_2$ is shown in Table 12.8. From the incremental unit cost values, a Monte Carlo Statistical analysis is performed using the procedure described in Section 16. From this analysis a most likely value of $57.77/lb $ThO_2$ and overall cost range of $77.61/lb $ThO_2$ to 39.95/lb $ThO_2$ is predicted for the unit cost of $ThO_2$ from the north block. Predictions for the central and south blocks are most likely values of $56.80/lb $ThO_2$ and $57.74/bl $ThO_2$ respectively with ranges of $87.98/bl $ThO_2$ to 30.96/lb $ThO_2$ and $89.20/lb $ThO_2$ to 31.70/lb $ThO_2$ respectively.

# TABLE 12.8. Uncertainty Ranges for Factors Affecting Bear Lodge Cost Estimates

| | North Block | | Central Block | | South Block | |
|---|---|---|---|---|---|---|
| | Most Likely Value | Range | Most Likely Value | Range | Most Likely Value | Range |
| Mine Capital Cost ($10$^3$/yr) | 7,041 | 10,561 to 6,337 | 7,041 | 10,561 to 6,337 | 7,041 | 10,561 to 6,337 |
| Unit Capital Cost ($/lb $ThO_2$) | 2.86 | 4.28 to 2.59 | 1.56 | 2.35 to 1.41 | 1.88 | 2.81 to 1.69 |
| Incremental Unit Cost ($/lb $ThO_2$) | | 1.42 to -.27 | | .79 to -.15 | | .93 to -.19 |
| Mine Operating Cost ($/10$^3$/yr) | 12,341 | 18,511 to 11,106 | 12,341 | 18,511 to 11,106 | 12,341 | 18,511 to 11,106 |
| Unit Capital Cost ($/lb $ThO_2$) | 5.00 | 7.51 to 4.50 | 2.75 | 4.11 to 2.47 | 3.29 | 4.93 to 2.96 |
| Incremental Unit Cost ($/lb $ThO_2$) | | 2.51 to -.50 | | 1.36 to -.28 | | 1.64 to -.33 |
| Mill Capital Cost ($/10$^3$/yr) | 56,133 | 84,199 to 50,520 | 56,133 | 84,199 to 50,520 | 56,133 | 84,199 to 50,520 |
| Unit Capital Cost ($/lb $ThO_2$) | 5.24 | 7.86 to 4.72 | 5.24 | 7.86 to 4.72 | 5.24 | 7.86 to 4.72 |
| Incremental Unit Cost ($/lb $ThO_2$) | | 2.62 to -.52 | | 2.62 to -.52 | | 2.62 to -.52 |
| Mill Operating Cost ($/10$^3$/yr) | 467,712 | 748,399 to 25,692 | 467,712 | 748,339 to 25,692 | 467,712 | 748,339 to 25,692 |
| Unit Capital Cost ($/lb $ThO_2$) | 43.67 | 69.87 to 24.02 | 43.67 | 69.87 to 24.02 | 43.67 | 69.87 to 24.02 |
| Incremental Unit Cost ($/lb $ThO_2$) | | 26.20 to -19.65 | | 26.20 to -19.65 | | 26.20 to -19.65 |
| Thorium Recovery Efficiency (%) | 60 | 40 to 85 | 60 | 40 to 85 | 60 | 40 to 85 |
| Recovery Efficiency Cost ($/lb $ThO_2$) | 57.80 | 85.62 to 43.57 | 53.20 | 79.82 to 39.89 | 54.10 | 81.15 to 40.57 |
| Incremental Unit Cost ($/lb $ThO_2$) | | 27.82 to -14.23 | | 26.62 to -13.31 | | 27.05 to -13.53 |

# REFERENCES

1.  M. H. Staatz, <u>Proceedings of The Uranium Industry Seminar</u>, 1978, GJO-108 (78) U.S. Dept. of Energy, Grand Junction, CO. pp. 109-115, 1978.

2.  F. J. Hurst, et al., <u>Estimated Costs for Recovery of Thorium and Uranium</u>, ORNL-3988, Oak Ridge National Laboratory, Oak Ridge, TN, November 1966.

3.  Jim F. Lemons, Jr. and Luis V. Coppa, <u>Mining and Processing Methods and Cost Models for the Recovery of Thorium from Domestic Occurrences</u>, GJBX-91(79), U.S. Dept. of Energy, Grand Junction, CO, pp. 14-17.

4.  S. R. Borrowman and J. B. Rosenbaum, <u>Recovery of Thorium from Ores in Colorado, Idaho, and Montana</u>, USBM 5916, United States Bureau of Mines, Update 1976, 16 pp.

## 13.0  THORIUM PRODUCTION FROM BALD MOUNTAIN, WYOMING

### 13.1 RESOURCE DESCRIPTION

Bald Mountain is located approximately 6 miles north of Sundance, Wyoming in the Bear Lodge Mountain area. With an elevation of 4,802 ft, the Bald Mountain deposit is entirely located in the Black Hills National Forest and is characterized by fairly steep hills and gullies with 800 ft to 1,000 ft relief. The deposit consists of a central core of precambrian granite with syenite porphyries of a tertiary age[1]. Microscopic examination of 1,400 lb of drill core from Bald Mountain showed the principal constituents are quartz and feldspars with lessor concentrations of garnet, hematite, limonite, magnetite, ilmenite, zircon, and monazite[2]. The monazite in 20 million tons of material is reported to average 2.5 lb/ton of ore. A small but high grade stratum continues 675,000 tons averaging 13.2 lb of monazite per ton[3]. For purposes of this study, however, the monazite is assumed to be evenly distributed throughout the rock. The average grade of the monazite is reported to be 8.8% $ThO_2$[2]. The average ore grade of the Bald Mountain deposit is assumed to be 0.013% or approximately 2.85 lb of monazite per ton of ore.

Bald Mountain contains approximately 2,600 tons of $ThO_2$. Other byproducts include as much as 7,700,000 tons of ilmenite and 16,000 tons of rare earth oxides[4].

### 13.2 MINE DESCRIPTION

Currently, Bald Mountain is not being mined for thorium. The conceptual Bald Mountain mine is based on the Palmer, Michigan mine assuming a similar deposit geometry. The conceptual mine shall produce approximately 10,000 tpd of ore assuming a mine life of 10 yr. The mine is designed to operate 3 shifts/day, five days/week for 240 day/yr.

#### 13.2.1  Mining Process

The mining process is the same for Bald Mountain mine as the Palmer, Michigan mine. The conceptual mining process is described in Section 11.2.1 of this report.

## 13.3 MILL DESCRIPTION

No commercial scale conglomerate benefication facilities currently exist; however, laboratory work indicates Bald Mountain monazite can be concentrated with 61% recovery[2].

A reference facility, based on this laboratory work, was designed to concentrate the output from a conceptual Bald Mountain mine. The mill has a capacity of 6,860 tpd of ore and produces 14 tons of concentrate. The concentrate contains 5.7% $ThO_2$ and 40.4% rare earth oxides. The mill is designed to operate three shifts per day, seven days per week, for 350 days per year. The mill is described in Section 11.3 of this report.

## 13.4 REFINERY DESCRIPTION

Monazite from Bald Mountain is shipped to a refining facility. The reference conglomerate refinery is described in Section 11.4 of this report.

## 13.5 BALD MOUNTAIN PRODUCTION COSTS

The production costs for developing an estimated 1,500 tons of $ThO_2$ from Bald Mountain is $45.70/lb $ThO_2$ (Table 13.1). After taking into account uncertainties in mine/mill capital and operating costs, refinery capital and operating costs, thorium recovery, plant capacity, and refinery location, a unit cost of $49.00/lb $ThO_2$ is estimated from the method described in Section 10. These costs are developed in the following sections.

TABLE 13.1. Bald Mountain Production Cost Summary

|  | Unit Cost ($/lb $ThO_2$) |
|---|---|
| Mining/Milling | $42.50 |
| Refining | 3.19 |
| Total from Estimates | $45.70 |
| Total Including Uncertainty | $49.00 |

### 13.5.1 Mine/Mill Production Cost

The production costs for operating the Hall Mountain mine are shown in Table 13.2. A fixed charge rate of .2668 is applied to the capital investment.

TABLE 13.2.  Bald Mountain Mine/Mill Production Costs

| | |
|---|---:|
| Annual Fixed Charges: | |
|    Depreciable Capital | $16,735,000 |
|     Non-Depreciable Capital | 728,000 |
| Annual Operating Costs | 15,335,000 |
| Subtotal | 32,798,000 |
| Depletion Allowance | 7,216,000 |
|    Annual Production Costs | 25,582,000 |
| Annual Levelized Production of $ThO_2$ | 602,000 |
| Unit Cost of Mining/Milling | $42.50/lb $ThO_2$ |

#### 13.5.1.1  Mine/Mill Capital Costs

Capital costs are calculated by scaling down the Palmer, Michigan cost estimate with a .75 scaling factor (Section 6.8). Capital costs for Bald Mountain are approximately $70,000,000. These costs are summarized in Table 13.3.

TABLE 13.3.  Bald Mountain Mine/Mill Capital Costs

| | |
|---|---:|
| Depreciable Capital Investment | $62,724,000 |
| Land and Working Capital | 7,276,000 |
|    TOTAL Capital Investment | $70,000,000 |

#### 13.5.1.2  Mine/Mill Operating Costs

Operating costs for Bald Mountain are also scaled down from Palmer using .75 scaling factor. Annual operating costs for Bald Mountain are estimated at $15,335,000.

## 13.5.2 Refinery Production Costs

The summary of refinery production costs are shown in Table 13.4. Bald Mountain ore constitutes only 8.6% of the total ore refined at the Palmer, Michigan refinery. For calculational purposes, capital and operating costs are scaled down linearly from the Palmer, Michigan costs to reflect the portion of the refinery with only Bald Mountain ore as input.

TABLE 13.4.  Bald Mountain Refinery Production Costs

Fixed Charges:

| | |
|---|---:|
| Depreciable Capital | $ 337,000 |
| Nondepreciable Capital | 19,000 |
| Annual Operating Costs | 729,000 |
| Subtotal | $1,085,000 |
| Annual Transportation Cost | 833,000 |
| Annual Production Cost | $1,918,000 |
| Annual Levelized $ThO_2$ Production | 602,000 |
| Unit Cost of Refining | $3.20/lb $ThO_2$ |

### 13.5.2.1 Transportation Cost

Transportation of thorium ore from the Bald Mountain area to Palmer would be provided by rail. Rail rates are estimated at $8.50/cwt. Annual transportation charges are estimated at $833,000.

## 13.5.3 Major Uncertainties

Major areas of uncertainty affecting the $ThO_2$ unit cost estimates include mine/mill capital and operating costs, refinery capital and operating costs, plant capacity, refinery location, and thorium recovery efficiency. Our estimates for the range of uncertainties in these areas are:

| | |
|---|---|
| Mine/Mill Capital Cost | +50% to -10% of calculated value |
| Mine/Mill Operating Cost | +50% to -50% of calculated value |
| Refinery Capital Cost | +50% to -10% of calculated value |
| Refinery Operating Cost | +50% to -10% of calculated value |
| Plant Capacity | from 5,000 to 20,000 tpd, with most likely value of 10,000 tpd |

13.4

| Refinery Location | onsite or at Palmer, Michigan, most likely at Palmer |
|---|---|
| Thorium Recovery Efficiency | from 42% to 82%, with a most likely value of 58% |

The impact of these uncertainty ranges on the unit cost of $ThO_2$ is shown in Table 13.5. From the incremental unit cost values, a Monte Carlo statistical analysis is performed using the procedure described in Section 16. From this analysis, a most likely value of $49/lb $ThO_2$ and overall cost range of $29/lb $ThO_2$ to $68/lb $ThO_2$ is predicted for the unit cost of $ThO_2$.

TABLE 13.5. Uncertainty Ranges for Factors Affecting $ThO_2$ Unit Cost

| | Most Likely Value | Range |
|---|---|---|
| Mine/Mill Capital Cost $10^3$/year | 13,621[a] | 20,432 to 12,259 |
| Unit Capital Cost $/lb $ThO_2$ | 22.63 | 33.94 to 20.36 |
| Incremental Unit Cost $/lb $ThO_2$ | | +11.31 to -2.27 |
| Mine/Mill Operating Cost $10^3$/year | 11,961[a] | 17,942 to 5,981 |
| Unit Operating Cost $/lb $ThO_2$ | 19.87 | 29.80 to 9.93 |
| Incremental Unit Cost $/lb $ThO_2$ | | +9.93 to -9.94 |
| Refinery Capital Cost $10^3$/year | 356 | 534 to 320 |
| Unit Capital Cost $/lb $ThO_2$ | .59 | .89 to .53 |
| Incremental Unit Cost $/lb $ThO_2$ | | +.30 to -.06 |
| Refinery Operating Cost $10^3$/year | 729 | 1.094 to 656 |
| Unit Operating Cost $/lb $ThO_2$ | 1.21 | 1.82 to 1.09 |
| Incremental Unit Cost $/lb $ThO_2$ | | +.61 to -.12 |
| Plant Capacity (tpd) | 10,000 | 5,000 to 20,000 |
| Unit Capacity Cost $/lb $ThO_2$ | 45.69 | 51.87 to 40.35 |
| Incremental Unit Cost $/lb $ThO_2$ | | +6.18 to -5.34 |
| Refinery Location | Palmer, Michigan | Palmer, Michigan or onsite |
| Unit Location Cost $/lb $ThO_2$ | 45.69 | 47.02 to 45.69 |
| Incremental Unit Cost $/lb $ThO_2$ | | +1.33 to 0 |
| Thorium Recovery Efficiency (%) | 58 | 42 to 82 |
| Unit Recovery Efficiency $/lb $ThO_2$ | 45.69 | 63.10 to 32.32 |
| Incremental Unit Cost $/lb $ThO_2$ | | +17.41 to -13.37 |

(a) Depletion allowance (22%) is subtracted from cost

# REFERENCES

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3.  V. R. Wilrath, and D. H. Johnson, Preliminary Reconnaissance Survey for Thorium, Uranium, and Rare-Earth Oxides, Bear Lodge Mountains, Crook County, Wyoming. Trace Elements Investigations Report 172, U.S. Geol. Survey, 26 pp., April 1953.

4.  R.H. Perry and C.H. Chilton, Chemical Engineers' Handbook, 5th Ed., McGraw-Hill Book Company, New York, N.Y., pp.25-16 and 25-17, 1973.

# 14.0 THORIUM PRODUCTION FROM CONWAY GRANITE

A study was conducted by the Atomic Energy Commission (AEC) in 1966 to determine the economics of obtaining thorium from Conway granite. This discussion is based on the AEC report[1,2], but the costs have been updated to 1978.

## 14.1 RESOURCE DESCRIPTION

Conway granite of New Hampshire is a large potential source of $ThO_2$. The deposit, occupying approximately 307 square miles in the White Mountains contains an estimated 35 million tons of thorium and extends to a depth of at least 1000 ft. Thorium is distributed evenly in the outcrop with an average grade of $56 \pm 6$ ppm (0.006%) $ThO_2$[1,2].

Typical Conway granite consists primarily of quartz, perthite, minor biotite, and plagioclase minerals. It is coarse-grained, pink, and massive although some outcrops are greenish and locally fine-grained. Uranium is associated with the thorium. Uranium content is at least one-fourth the thorium content.

## 14.2 MINE DESCRIPTION

Granite is mined by open pit methods. Mine production is assumed to be 100,000 tons/day of ore and approximately 20,000 tons/day of waste. The mine operates 3 shifts a day, 350 days/year for 20 years. Ore recovery is assumed to be 100%.

### 14.2.1 Mining Process

The mining process is assumed to be similar to the open pit methods utilized at Palmer, Michigan (see Section 11.2.1). The Conway mining process utilizes standard drills of the "down the hole" type to make blast holes. Shovels (8 $yd^3$) load the ore and waste into electric wheel trucks. The ore is carried to the mill. The waste is hauled by electric train to a disposal area.

## 14.3 MILL DESCRIPTION

Granite is not currently processed for thorium in the United States. However, laboratory studies on the benefication of thorium and uranium from granite reveal that direct leaching of the ore with dilute sulfuric acid yields the best results. The reference granite mill is assumed to process 100,000 tpd of ore and produce 7,200 lb of $ThO_2$ in the form of 6.3 tpd of concentrate assuming a thorium recovery of 60%. This concentrate contains 57% $ThO_2$ and 16.7% uranium. The mill operates 3 shifts/day, 350 days/year for 20 yr.

### 14.3.1 Milling Process

The milling circuit consists of acid leaching followed by amine extraction and ammonia precipitation. The milling process is summarized in the following sections. A process flow diagram is shown in Figure 14.1.

#### 14.3.1.1 Crushing and Grinding

The mined ore is crushed to 8 in. in large gyratory crushers. The crushed ore is transported to an outside stockpile holding about an 8 hr supply of granite. The ore is then fed to dry, autogenous mills of the aerofall type for grinding to minus 20 mesh.

#### 14.3.1.2 Acid Leaching

The ground ore is then agglomerated in rotating drums with 20-60 lb of sulfuric acid per ton of ore to produce a mix containing 85% solids. The agglomerated ore is then conveyed to concrete vats and cured for 1 to 5 hours. Recycle process solutions are utilized to wash the ore in the vats to obtain a thorium and uranium solution. The relatively small, high acid portion of the wash solution obtained first is recycled to the agglomeration operation. The next portion of the wash cycle constitutes the pregnant liquor and goes to the solvent extraction step. The latter portion of the wash solution with a much lower concentration of thorium and uranium is recycled for washing subsequent batches of ore. A final flush solution containing ore tailings and water goes to lime neutralization and waste.

# CONWAY GRANITE MILL

ORE



FIGURE 14.1.  Conway Granite Refinery

## FLOW STREAMS FOR CONWAY GRANITE MILL

1.  4545.45 tons/hr ore w/$^{232}$Th = 480 lb/hr
2.  454.5 lb/hr w/$^{232}$Th = 2.4024 x 10$^{-3}$ lb/hr
3.  4545.22 tons/hr w/$^{232}$Th = 480 lb/hr
4.  800 tons/hr
5.  72.72 tons/hr $H_2SO_4$
6.  5345.22 tons/hr w/$^{232}$Th = 480 lb/hr
7.  1181.7 tons/hr $H_2O$
8.  727.28 tons/hr
9.  5799.46 tons/hr w/$^{232}$Th = 167.96 lb/hr
10. 2908.8 tons/hr
11. 2908.98 tons/hr w/$^{232}$Th = 312.04 lb/hr
12. 500 tons/hr amine solvent
13. 500.18 tons/hr w/$^{232}$Th = 312.04 lb/hr
14. 63.63 tons/hr $Na_2CO_3$
15. 63.81 tons/hr w/$^{232}$Th = 312.04 lb/hr
16. 2.73 tons/hr $H_2SO_4$
17. 909 lb/hr $NH_3$
18. 66.71 tons/hr w/$^{232}$Th = 23.999 lb/hr
19. 574.24 lb/hr w/$^{232}$Th = 288.04 lb/hr
20. 0.029 lb/hr w/$^{232}$Th = 1.452 x 10$^{-2}$ lb/hr
21. 574.21 lb/hr w/$^{232}$Th = 288.024 lb/hr

FIGURE 14.1. (cont'd)

### 14.3.1.3 Solvent Extraction

The pregnant liquor goes to a two-stage solvent extraction section.
The solvent consists of .01 M 1-nondecylamine and .01 M N-benzyl-1-(3
ethylpentyl)-4-ethyloctyl amine in 99% kerosene-1% tridecanol diluent.
Thorium and uranium are then costripped from the extract with .25 M $Na_2CO_3$
solution in two stages. Standard mixer-settler equipment performs the ex-
traction and stripping operations.

### 14.3.1.4 Packaging

The thorium-uranium precipitate is filtered, dried, and stored for further refinement and separation.

### 14.4 REFINERY DESCRIPTION

The reference granite refinery shall be similar to the reference stockpile refinery except for the addition of a extraction cascade which precedes the $ThO_2$ extraction steps (see Figure 14.2). The cascade shall utilize 10 stages and a 5% TBP solvent to remove uranium from the thorium solution. The refining process is described in Section 7.2 of this report. The uranium recovery steps shall be discussed below.

### 14.4.1 Uranium Recovery

The thorium-uranium solution enters a 5 stage solvent extraction section where the aqueous is contacted with a 5% solution of TBP in kerosene. The thorium-bearing aqueous continues on to the next solvent extration section. The organic solution enters a 5-stage scrub section where the uranium is removed from the solvent using a 4 N nitric acid solution. The uranium solution is precipitated with ammonia, dewatered, and dried. The stripped solvent is recycled.

### 14.5 CONWAY GRANITE PRODUCTION COST

The production costs for developing $ThO_2$ from Conway granite are summarized in Table 14.1. After accounting for uncertainties in mine, mill, and refinery capital and operating costs, byproduct credit, plant capacity, and thorium recovery efficiency, a most likely cost of \$59.30/lb $ThO_2$ was determined by the method described in Section 16. These costs are developed in the following sections.

### 14.5.1 Mine/Mill Production Costs

The production costs for operating the Conway granite mine and mill are shown in Table 14.2. A fixed charge of .2330 was applied to annualize non-depreciable capital investment over 30 yr.

FIGURE 14.2. Conway Granite Refinery

1. 570 lb/hr w/$^{232}$Th = 285.9 lb/hr
2. 299 lb/hr $HNO_3$
3. 491 lb/hr $H_2O$
4. 1360 lb/hr w/$^{232}$Th = 285.9 lb/hr
5. 663 lb/hr w/$^{232}$Th = 5.72 lb/hr
6. 697 lb/hr w/$^{232}$Th = 280.18 lb/hr
7. 1367 lb/hr 4M $HNO_3$
8. 1367 lb/hr w/$^{232}$Th = .308 lb/hr
9. 33 lb/hr $NH_3$
10. 1400 lb/hr w/$^{232}$Th = .308 lb/hr
11. 1216 lb/hr w/$^{232}$Th = .308 lb/hr
12. 184 lb/hr
13. 56.86 lb/hr $H_2O$
14. 127.14 lb/hr (70% $U_3O_8$)
15. 697 lb/hr w/$^{232}$Th = 279.87 lb/hr
16. 557 lb/hr $HNO_3$
17. 1254 lb/hr w/$^{232}$Th = .264 lb/hr
18. 6359 lb/hr
19. 6677 lb/hr w/$^{232}$Th = 279.606 lb/hr
20. 6677 lb/hr of .1 N $HNO_3$
21. 318 lb/hr
22. 6677 lb/hr w/$^{232}$Th = 279.606 lb/hr
23. 5351 lb/hr $H_2O$
24. 1326 lb/hr w/$^{232}$Th = 279.606 lb/hr
25. 7.4 lb/hr $HNO_3$
26. 87 lb/hr $H_2O$
27. 112.6 lb/hr oxalic acid
28. 1599.6 lb/hr w/$^{232}$Th = 279.606 lb/hr
29. 288.9 lb/hr $H_2O$
30. 1070.9 lb/hr w/$^{232}$Th = 2.29 lb/hr
31. 309.7 lb/hr $Na_2CO_3$
32. 1380.6 lb/hr w/$^{232}$Th = 2.29 lb/hr
33. 817.6 lb/hr w/$^{232}$Th = 277.316 lb/hr
34. 326 lb/hr $H_2O$
35. 581.6 lb/hr w/$^{232}$Th = 277.316 lb/hr
36. 43.3 lb/hr $O_2$
37. 100.8 lb/hr $H_2O$ and 205.8 lb/hr $CO_2$ w/$^{232}$Th = 5.72 lb/hr
38. 5.66 lb/hr $^{232}$Th
39. 307.7 lb/hr w/$^{232}$Th = 5.72 x $10^{-2}$ lb/hr
40. 311.8 lb/hr w/$^{232}$Th = 271.60

<u>FIGURE 14.2.</u>  (cont'd)

14.7

TABLE 14.1. Conway Granite Production Costs

|  | Unit Cost $/lb |
|---|---|
| Mining/Milling | $ 58.70 |
| Refining | 3.30 |
| Total Product Cost (No Byproducts) | $ 62.00 |
| Uranium Byproduct Credit | 11.70 |
| Total Product Cost (Full Byproducts) | $ 50.20 |
| Total (Including Uncertainty) | $ 59.30 |

TABLE 14.2. Conway Granite Mine/Mill Production Costs

| Annual Fixed Charges: | |
|---|---|
| Depreciable Capital | $ 73,683,000 |
| Nondepreciable Capital | 3,347,000 |
| Annual Operating Costs (Mine) | 44,940,000 |
| Annual Operating Costs (Mill) | 91,922,000 |
| Annual Production Costs | $213,892,000 |
| Depletion Allowance | 47,056,000 |
| Annual Production Cost | $166,836,000 |
| Average Levelized Production of $ThO_2$ | 2,842,700 lb |
| Unit Cost of Mining and Milling | $58.70/lb $ThO_2$ |

### 14.5.1.1 Mine/Mill Capital Costs

Capital costs were estimated for the mine and mill in two ways: (1) by escalation of the AEC study and (2) by scaling up of Palmer, Michigan mine/mill capital costs. The costs for Palmer, Michigan do not include a solvent extraction facility. However, land costs are probably higher than necessary because of purchasing the town of Palmer. These costs are shown in Table 14.3. An average cost of $348,622,000 is used in this study.

### 14.5.1.2 Mine/Mill Operating Costs

Mine operating costs are estimated from the Palmer, Michigan, study to be $1.07/ton of ore or approximately $44,940,000 annually.

14.8

TABLE 14.3. Conway Granite Mine/Mill Capital Cost

|  | AEC | Palmer |
| --- | --- | --- |
| Total Depreciable Capital Investment | $ 284,466,000 | $ 345,840,000 |
| Working Capital | 26,823,000 | 40,115,000 |
| Total Capital Investment | $ 311,289,000 | $ 385,955,000 |

Mill operating costs were determined assuming labor, reagent and utility requirements from the AEC study. Other costs were determined using the procedure outlined in Section 6.0.

Mill operating costs are approximately $91,922,000 annually (Table 14.4). Reagent and utility requirements were derived from the AEC study and are shown in Tables 14.5 and 14.6.

14.5.2 Refinery Production Costs

The unit cost of refining is estimated from the sludge refining data in Table 7.1 by assuming the concentrate contains 57% $ThO_2$. By interpolating between costs for processing 46% and 88% $ThO_2$, the refining cost for Conway granite is $3.01.

TABLE 14.4. Mill Operating Costs

| | |
| --- | --- |
| Operating Labor (87 men) | $ 5,890,000 |
| Direct Supervisory and Clerical | 884,000 |
| Maintenance and Repairs | 10,000,000 |
| Utilities | 11,750,000 |
| Reagents | 57,422,000 |
| Operating Supplies | 1,500,000 |
| Laboratory Charges | 900,000 |
| Plant Overhead | 10,060,000 |
| Administrative Overhead | 2,516,000 |

TABLE 14.5. Reagent Consumption and Expense

| Reagent | Annual Consumption | Unit Cost | Annual Cost |
|---|---|---|---|
| Sulfuric Acid | 700,000 tons | $ 55/ton | $38,500,000 |
| Lime | 434,000 tons | 37/ton | 16,058,000 |
| Soda Ash | 13,125 tons | 68.9/ton | 904,000 |
| Ammonia | 3,500 tons | 120/ton | 420,000 |
| Amine Solvents | 420,000 lbs | 1/ton | 420,000 |
| Total Solvents | 2,800,000 gal | 0.4 gal | 1,120,000 |
| | | | $57,422,000 |

TABLE 14.6. Utilities

| | | |
|---|---|---|
| Electricity | (10.7 kWh/ton of ore) | $11,235,000 |
| Water | (12,500 gpm) | 312,000 |
| Fuel Oil | (0.008/ton of ore) | 280,000 |
| | | $11,830,000 |

### 14.5.3 Byproduct Credit

Byproduct credit is calculated assuming $40/lb contained $U_3O_8$. Assuming a $U_3O_8$ production of 771,540 lb/yr for a 600 lb refinery, byproduct revenue is approximately $30,861,600 annually. After subtracting the incremental uranium production costs, full byproduct credit is estimated at $29,403,000 annually. This is approximately $11.70/lb $ThO_2$.

#### 14.5.3.1 Byproduct Capital Cost

The estimated additional capital cost associated with recovering uranium is summarized in Table 14.7. This cost is approximately $1,074,000.

#### 14.5.3.2 Byproduct Operating Cost

The additional operating costs for byproduct recovery are summarized in Table 14.8. Labor costs are based on the number of operators necessary to equipment operation. Supervision, maintenance and repairs, operating supplies, laboratory charges, utilities, and indirect costs are estimated using the procedure described in Section 6.0. Reagent expense is itemized in Table 14.9.

14.10

## TABLE 14.7. Conway Granite Byproduct Capital Cost

| | |
|---|---:|
| Byproduct Buildings & Equipment Cost | $ 745,000 |
| Contingency (15% of DPC) | 108,000 |
| Feasibility Study | 100,000 |
| Total Depreciable Capital Investment | 953,000 |
| Land | 3,000 |
| Working Capital | 108,200 |
| | $1,074,000 |

## TABLE 14.8. Conway Granite Byproduct Operating Cost

**Direct Costs**

| | |
|---|---:|
| Operating Labor (5 men/shift) | $ 339,000 |
| Supervision | 51,000 |
| Maintenance Repairs | 64,000 |
| Operating Supplies | 10,000 |
| Utilities (5% of direct costs)[3] | 44,000 |
| Laboratory Charges | 51,000 |
| Reagents | 325,000 |

**Indirect Costs**

| | |
|---|---:|
| Plant Overhead | $ 272,000 |
| Administrative Overhead | 68,000 |
| | $1,224,000 |

## TABLE 14.9. Reagent Consumption and Expense

| Reagent | Annual Consumption | Unit Cost | Annual Cost |
|---|---|---|---|
| $HNO_3$ | 1,400 tons | $10.50/100 lb | $294,000 |
| $NH_3$ | 150 tons | 120.00/ton | 18,000 |
| TBP | 8,000 lb | 0.88/lb | 7,000 |
| Kerosene | 15,000 gal | 0.40/gal | 6,000 |
| | | | $325,000 |

## 14.5.3.3 Byproduct Production Cost

Production costs for uranium byproducts are summarized below. These costs are approximately $1,459,000 annually. Annual revenue is estimated at $30,862,000 assuming a processing rate of 600 lb/hr of concentrate.

| | |
|---|---|
| Depreciable Fixed Charge | $ 223,000 |
| Fixed Charge on Land and Working Capital | 12,000 |
| Operating Costs | 1,224,000 |
| Production Cost | $1,459,000 |

## 14.6 MAJOR UNCERTAINTIES

Major areas of uncertainty affecting the $ThO_2$ unit cost estimates include mine/mill capital cost, mine, mill, and refinery capital and operating costs, byproduct credit, plant capacity, and thorium recovery efficiency. Estimates for the ranges of uncertainties in these areas are:

| | |
|---|---|
| Mine/Mill Capital Cost | +50% to -10% of calculated value |
| Mine/Mill Operating Cost | +50% to -10% of calculated value |
| Refinery Capital Cost | +50% to -10% of calculated value |
| Refinery Operating Cost | +50% to -10% of calculated value |
| Byproduct Credit | full credit to no credit with full credit most likely |
| Plant Capacity | 50,000 to 150,000 tpd with most likely value of 100,000 tpd |
| Thorium Recovery Efficiency | 32% to 74% with most likely value of 57% |

The impact of these uncertainty ranges on the unit cost of $ThO_2$ is shown in Table 14.10. From the incremental unit cost values, a Monte Carlo statistical analysis is performed using the procedure described in Section 16. From this analysis, a most likely value of $59.30/lb $ThO_2$ and overall cost range of $36.70 to $93.70/lb $ThO_2$ is predicted for the unit cost of $ThO_2$.

TABLE 14.10.  Uncertainty Ranges for Factors Affecting $ThO_2$ Unit Cost

| | Most Likely Value | Range |
|---|---|---|
| Mine/Mill Capital Cost | 60,083[a] | 90,125 to 54,075 |
| Unit Capital Cost | 21.13 | 30.70 to 19.02 |
| Incremental Unit Cost | | +10.57 to -2.11 |
| Mine Operating Cost $10^3$/year | 35,053[a] | 52,580 to 31,548 |
| Unit Operating Cost $/lb $ThO_2$ | 12.33 | 18.49 to 11.10 |
| Incremental Unit Cost $/lb $ThO_2$ | | +6.16 to -1.23 |
| Mill Operating Cost $10^3$/year | 71,699[a] | 107,549 to 64,529 |
| Unit Operating Cost $/lb $ThO_2$ | 25.22 | 37.83 to 22.70 |
| Incremental Unit Cost $/lb $ThO_2$ | | +12.61 to -2.52 |
| Unit Refinery Capital and Operating Cost $/lb $ThO_2$ | 3.01 | 4.52 to 2.71 |
| Incremental Unit Cost $/lb $ThO_2$ | | +1.51 to -.30 |
| Byproduct Credit % | 100 | 100 to 0 |
| Unit Credit Cost $/lb $ThO_2$ | -11.75 | -11.75 to 0 |
| Incremental Unit Cost $/lb $ThO_2$ | | 0 to +11.75 |
| Plant Capacity tpd | 100,000 | 150,000 to 50,000 |
| Unit Capacity Cost $/lb $ThO_2$ | 50.25 | 45.17 to 59.75 |
| Incremental Unit Cost $/lb $ThO_2$ | | -5.08 to +9.50 |
| Thorium Recovery Efficiency % | 57 | 32 to 74 |
| Recovery Efficiency Cost $/lb $ThO_2$ | 50.25 | 89.51 to 38.71 |
| Incremental Unit Cost $/lb $ThO_2$ | | +39.26 to -11.54 |

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(a) Depletion allowance (22%) is subtracted from cost

# REFERENCES

1.  Hurst, F. J., Crouse, D. J., and Brown, K. B., Recovery of Thorium and Uranium From Granitic Rocks. ORNL-3987, Oak Ridge National Laboratory, Oak Ridge, TN, October 1966.

2.  Hurst, F. J., et al., Estimated Costs for Recovery of Thorium and Uranium. ORNL-3988, Oak Ridge National Laboratory, Oak Ridge, TN, November 1966.

3.  Merritt, R. C., The Extractive Metallurgy of Uranium. Colorado School of Mines, Golden, CO, 1971.

## 15.0 PIEDMONT PLACER DEPOSITS

### 15.1 RESOURCE DESCRIPTION

The United States Geological Survey[1] estimates that there are about 784,000 short tons of monazite, a principal source of thorium in placer deposits located in the Inner Piedmont Belt between the Savannah River in South Carolina and the Catawba River in North Carolina. Fluviatile placers in flood plains constitute the main source of monazite in the area. Placers also exist north and south of this belt, but the monazite content was found to be sparse.

The USGS identified 84 placer deposits downstream from the river headwaters in broad, U-shaped valleys. These placers range from $1 \times 10^6$ yd$^3$ to $10 \times 10^6$ yd$^3$ of alluvium with an average of $3.1 \times 10^6$ yd$^3$. The deposits are an average of 14 to 16 feet thick and contain an average of 1.3 lb/yd$^3$ of monazite. From these figures, the placers are estimated to contain about 375,000 tons of monazite which is contained mainly in the sandy gravel strata of the placer deposit.

Other smaller, sparser placers also exist in the area. An additional 5,000 small placers deposits are located at the headwaters in narrow V-shaped valleys. These deposits contain an estimated 20,000 yd$^3$ of aluvium per deposit with tenor of 5 lb/yd$^3$ of monazite. These deposits account for approximately 250,000 short tons of monazite mineral. Monazite is widely disseminated in the fine sediments downstream of gravel-bearing placers. The average tenor of these sediments is about .04 lb/yd$^3$ of monazite. These sediments appear impractical to mine for thorium within the time frame of this study.

Over the time frame of this study, it is concluded that only the largest fluviatile placers in the flood plains deposits could be mined. Utilizing current mining technology, recovery efficiency is only about 58%. Thus, the total amount of monazite available from Piedmont placer deposits is about 217,500 short tons.

The ThO$_2$ content in the monazite of these deposits ranges from 5.27% to 5.83% with an average of 5.6%. The rare earth oxide content is 62.8% and

the $U_3O_8$ content ranges from .18% to .98% with an average between .38% and .52%. Heavy minerals other than monazite occurring in these placer deposits are as follows:

| Heavy Mineral | Tenor lb/yd$^3$ Aluvium |
|---------------|-------------------------|
| Ilmenite | 5.3 |
| Rutile | .3 |
| Zircon | .6 |

The amount of reactor grade thorium that would be produced from the largest fluviatile placers is calculated to be 11,000 short tons of $ThO_2$. In addition, these deposits would also yield approximately 123,000 tons of rare earth oxides, 400,000 tons of ilmenite, 23,000 tons of rutile, and 45,000 tons of zircon.

## 15.2 PRELIMINARY MINE DESCRIPTION

Since the Piedmont Placer deposits are remote and small in size, mine equipment must be relocated several times. For this reason, the conceptual mine model uses dragline (doodlebug) dredge methods. Dragline dredging is currently employed to recover precious minerals such as gold, tin, and diamonds. The equipment is especially adapted to mining in remote districts where water is at a premium. Advantages of dragline dredging over conventional dredge methods used currently for heavy mineral recovery include: (1) lower capital equipment cost, (2) greater ease of transporting equipment, and (3) smaller dredge pond requirements.

Dragline dredging is accomplished by 2 cubic yard crawler mounted draglines operating on dry land. These draglines feed a dredge wet mill which processes approximately 200 tph of placer sand and discharges 675 lb/hr of heavy mineral concentrate to a stockpile adjacent to the dredge pond. The concentrate is dewatered and transported by truck to a dry mill.

The dry mill separates monazite from the heavy mineral concentrate by gravity and magnetic separation. To mine Piedmont Placers, approximately 15 dragline wet mill units will feed into the dry mill. The dry mill will

process 56 tons per day of concentrate and produces 6.91 ton/day of a 95% monazite sand product. This concentrate is shipped to an extraction and refining facility similar to those described in a report by Enderlin[2].

The draglines and dredge are moved on an average interval of 2.5 years. The parts are dismantled and loaded into a truck for transport to a new deposit. The plant can be dismantled and reassembled in about one week.

## 15.3 PRELIMINARY COST ESTIMATE

A detailed cost estimate for Piedmont Placers was not developed since the preliminary estimate of mining costs exceeded $142/lb $ThO_2$. Assuming full byproduct credit for ilmenite, rutile, and zircon would reduce the cost to $140/lb $ThO_2$. Thus, the cost of thorium from the Piedmont Placers would exceed the cost of thorium from the most expensive resource studied - Conway Granite. Because of the large reserve contained in the Conway Granite, we conclude that the Piedmont Placers will not be mined within the timeframe of this analysis.

# REFERENCES

1.  W. C. Overstreet, et al., _Fluvial Monazite Deposits in the Southeastern United States_, U.S. Geological Survey Professional Paper 568, 1968, p. 85, plates in pocket.

2.  W. I. Enderlin, _An Assessment of U.S. Domestic Capacity for Producing Reactor Grade $ThO_2$ and Controlling Associated Wastes and Effluents_, Battelle Northwest Laboratories, PNL-2593, February 1978.

# 16.0 UNCERTAINTY ANALYSIS

## 16.1 INTRODUCTION

In estimating the production cost for $ThO_2$ from each resource, we first established the "most likely" estimates for each major variable, e.g., ore grade, capital cost recovery rate, etc. We called these the "reference conditions". We then determined the production cost under the reference conditions.

However, there is considerable uncertainty surrounding the true value for each variable, particularly in a study of this nature. For example, although the most likely value for ore grade is estimated to be 1%, the uncertainty is such that the true value might lie anywhere in a range from 0.5 to 2.0%. In this case the probability of the true value occuring at a specific point in this range can only be expressed through a statistical distribution. Since a similar statistical distribution is applied to many of the major variables in this study, we used a computer code, OPS*, to calculate production costs based on the joint probabilities of occurrence at specific values.

## 16.2 COMPUTER ANALYSIS

OPS calculates the distribution of production costs for reactor grade $ThO_2$ based on high, most likely, and low estimates of cost variables. In order to illustrate the OPS procedure for determining price variation, the Palmer, Michigan uncertainty analysis will be discussed. In this case an open pit mine is designed to produce 10,000 tons per day of ore which is concentrated and refined to reactor grade $ThO_2$. The major cost variables are as follows:

1. Mine/mill operating costs
2. Mine/mill capital costs
3. Refinery operating costs

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* R. W. Legan and R. L. Engel, OPS: Risk Analysis in Capital Investment and Development Opportunities, BNWL-1562, May 1972.

4. Refinery capital costs
5. Ore recovery
6. Byproduct credit

For each of the variables the high, most likely, and low values are estimated (Table 16.1). These values are then translated into the equivalent $ThO_2$ production costs as described in Section 6.0. Items 1 through 4 are costs based on the most likely thorium recovery. Item 5 accounts for cost differences based on the variability in ore recovery.

TABLE 16.1. Uncertainty Ranges $ThO_2$ Unit Cost of Palmer, Michigan

| | Most Likely Value | Range |
|---|---|---|
| Mine/Mill Capital Cost $10^3$/year | 35,034 | 52,551 to 31,530 |
| Unit Cost $/lb $ThO_2$ | 5.45 | 8.17 to 4.90 |
| Mine/Mill Operating Cost $10^3$/year | 34,903 | 52,354 to 31,412 |
| Unit Cost $/lb $ThO_2$ | 5.43 | 8.14 to 4.88 |
| Refinery Capital Cost $10^3$/year | 4,152 | 6,228 to 3,737 |
| Unit Cost $/lb $ThO_2$ | .65 | .97 to .58 |
| Refinery Operating Cost $10^3$/year | 8,519 | 12,779 to 7,667 |
| Unit Cost $/lb $ThO_2$ | 1.32 | 1.99 to 1.19 |
| Ore Recovery (%) | 56 | 50 to 62 |
| Unit Cost $/lb $ThO_2$ | 12.88 | 14.43 to 11.63 |
| Byproduct Credit | No Credit | Up to $12 Million/yr |
| Unit Cost $/lb $ThO_2$ | 0 | 0 to -1.87 |

Since the probability distribution for these variables is not known, OPS assigns a beta probability distribution to each cost item. The Monte Carlo simulation method is utilized to select a single random value from each distribution. The Monte Carlo method is such that the most likely values are generated more often than unlikely values, but the specific outcome of any simulation is unpredictable. The cost simulation is done over and over again to obtain a statistically significant distribution of possible

16.2

costs for $ThO_2$. Other output from the code includes:

1. Number of simulations
2. Average cost
3. Standard deviation
4. Maximum value observed
5. Minimum value observed
6. Skewness estimate
7. Kurtosis estimate

Skewness is a measure of assymetry. If this value is less than zero, the distribution is skewed to left; greater than zero, the distribution is skewed to the right.

Kurtosis is a measure of peakedness. If this value is greater than 3.0, the distribution is more peaked than a standard distribution; if less than 3.0, the distribution is less than a normal distribution. A flat distribution has a value of 1.8.

The output for Palmer, Michigan is shown in Table 16.2.

TABLE 16.2. Uncertainty Analysis for Palmer, Michigan Deposit

| | |
|---|---:|
| Number of Entries in Table | 200 |
| Average | 13.34 |
| Standard Deviation | 1.76 |
| The Maximum Value is | 16.96 |
| The Minimum Value is | 9.72 |
| Skewness Estimate from Moments | -.00935 |
| Skewness Estimate from K-Statistics | -.00944 |

The distribution is skewed to the left; that is, it has a left (low) tail.

| | |
|---|---:|
| Kurtosis Estimate from Moments | 2.30 |
| Kurtosis Estimate from K-Statistics | 2.68 |

The distribution is less peaked than a normal distribution. A value of 1.8 is a flat distribution.

16.3

Data indicates the expected cost for Palmer, Michigan thorite is between $17.00 and $9.70. The average cost is $13.30. The standard deviation is $1.80. This means that 68.3% of the time the price of thorium would range from $11.58-$15.10.

OPS also prints a histogram of the distribution curve. The data in Table 16.2 are illustrated in Figure 16.1. The average (mean) cost represented by the letter "A" is approximately $13.50, but the most likely (modal) cost is $14.20.

```
 9.900+************
10.100+****
10.300+**
10.500+**
10.700+******
10.900+*********************
11.100+**
11.300+****************
11.500+*******
11.700+***********
11.900+***************
12.100+*******
12.300+*********
12.500+*************
12.700+*******************
12.900+****************************
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14.900+********************
15.100+**********
15.300+******************
15.500+
15.700+**********
15.900+***
16.100+**************
16.300+************
16.500+****
16.700+**********
16.900+
17.100+****
```

FIGURE 16.1. Distribution of Unit Cost for Palmer, Michigan

16.4

Table 16.3 is a frequency table. The table presents for several intervals of calculated values:

1. The frequency
2. The relative frequency (percent)
3. Cumulative percentages
4. Cumulative remainder (100 minus item 3)

TABLE 16.3. Frequency Distribution for Palmer, Michigan

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
9.90	6	3.00	3.00	97.00
10.10	2	1.00	4.00	96.00
10.30	1	.50	4.50	95.50
10.50	1	.50	5.00	95.00
10.70	3	1.50	6.50	93.50
10.90	10	5.00	11.50	88.50
11.10	1	.50	12.00	88.00
11.30	8	4.00	16.00	84.00
11.50	3	1.50	17.50	82.50
11.70	5	2.50	20.00	80.00
11.90	7	3.50	23.50	76.50
12.10	3	1.60	25.00	75.00
12.30	4	2.00	27.00	73.00
12.50	6	3.00	30.00	70.00
12.70	9	4.50	34.50	65.50
12.90	12	6.00	40.50	59.50
13.10	7	3.50	44.00	56.00
13.30	9	4.50	48.50	51.50
13.50	4	2.00	50.50	49.50
13.70	9	4.50	55.00	45.00
13.90	8	4.00	59.00	41.00
14.10	13	6.50	65.50	34.50
14.30	13	6.50	72.00	28.00
14.50	0	.00	72.00	28.00
14.70	9	4.50	76.50	23.50
14.90	9	4.50	81.00	19.00
15.10	4	2.00	83.00	17.00
15.30	8	4.00	87.00	13.00
15.50	0	.00	87.00	13.00
15.70	4	2.00	89.00	11.00
15.90	1	.50	89.50	10.50
16.10	7	3.50	93.00	7.00
16.30	6	3.00	96.00	4.00
16.50	2	1.00	97.00	3.00
16.70	4	2.00	99.00	1.00
16.90	0	.00	99.00	1.00
17.10	2	1.00	100.00	.00

16.5

The cumulative percentage estimates the probability that the calculated attribute will be less than a certain amount. The cumulative remainder tells us the probability that the calculated value will be more than a certain amount. Thus, for Palmer, Michigan, there is a 5% chance that ThO_2 will cost less than $9.90 or cost more than $17.10.

16.3 RESULTS

The uncertainty analysis method utilized for Palmer, Michigan was employed for all deposits. The results are shown in Table 16.4. Frequency tables for each deposit are shown in Appendix F.

TABLE 16.4. Range of Unit Costs for Thorium Oxide

Deposit	Most Likely Value	High Value	Low Value
Stockpiles			
Fernald	2.80	4.20	2.00
Miamisburg Curtis Bay	4.10	5.60	2.90
Chattanooga	5.80	8.50	4.30
Lemhi Pass (Stage 1)	15.70	22.90	10.80
Lemhi Pass (Stage 2)	12.10	17.40	8.90
Hall Mountain	6.30	9.30	5.60
Wet Mountain	8.30	13.30	5.90
Palmer	14.20	17.10	9.70
Bald Mountain	49.00	68.00	29.00
Bear Lodge	57.40	84.90	34.20
Conway Granite	59.30	88.20	38.00

17.0 DOMESTIC THORIUM SUPPLY CURVE

17.1 DESCRIPTION

The supply curve illustrates that sufficient amounts of thorium exist to supply a domestic thorium-reactor economy. Most likely costs of production range from $3 to $60/lb ThO_2. Costs including undertainty range from $2 to $88/lb ThO_2. Near-term thorium oxide resources include the stockpiles in Ohio, Maryland, and Tennessee and the thorite deposits at Hall Mountain, Idaho. Costs are under $10/lb thorium oxide. Longer term economic deposits include Wet Mountain, Colorado; Lemhi Pass, Idaho; and Palmer, Michigan. Most likely costs are under $20/lb thorium oxide. Long-term deposits include Bald Mountain, Wyoming; Bear Lodge, Wyoming; and Conway, New Hampshire. Costs approximately equal or exceed $50/lb thorium oxide.

The supply curve for thorium oxide was developed for domestic thorium resources (see Figure 17.1). The curve illustrates the relationship between

SUPPLY OF THORIUM OXIDE, SHORT TONS OF THO_2

FIGURE 17.1A. Domestic Thorium Supply Curve

17.1

FIGURE 17.1B. Domestic Thorium Supply Curve

production costs and quantity of thorium oxide available. All production costs are in 1978 dollars. Production cost figures include costs for baseline environmental control. The effect of more stringent environmental control measures will be developed in a subsequent report.

In the supply curve, the quantity of reserves is illustrated for each deposit as a function of the most likely production cost by a rectangular area on the supply curve. A double arrow drawn perpendicular to the top of the rectangle illustrates high and low cost estimates for each resource. Resources are arrayed by plotting production costs in increasing order except for Lemhi Pass. The Lemhi Pass resource shows a decrease in cost because Lemhi Pass is mined in two stages. The first stage of mining is higher because a lower grade of ore is developed. As mining progresses, higher grades of ore are accessed (second stage) and production costs decrease.

The supply curve shows production costs for proven reserves only unless data is available to allow cost estimates for probable and potential resources.

17.2

Data used to develop the supply curve is derived from the production costs and the quantity of reserves developed which are summarized in Table 17.1. The total estimated amount of reserves are also shown. To the extent that these additional reserves can be developed, the supply curve would shift downward.

TABLE 17.1. Summary of Development Costs for Domestic ThO_2 Resources

Resource	Production Costs Most Likely	Range	Quantity Recovered Short Tons	Quantity in Deposit Short Tons [a]
Fernald	5.80	$ 4.30 - $ 8.50	≈1,500	1,500
Miamisburg	4.10	$ 2.90 - $ 5.60	≈800	800
Curtis Bay	4.10	$ 2.90 - $ 5.60	≈400	400
Chattanooga	5.80	$ 4.30 - $ 8.50	≈360	360
Lemhi Pass	13.90 [b]	$22.90 - $ 8.90	23,000	187,700
Hall Mountain	6.30	$ 5.60 - $ 9.30	≈5,000	35,000
Wet Mountain	8.30	$ 5.90 - $13.30	61,000	224,900
Palmer	14.20	$ 9.90 - $17.10	70,000	193,000
Bear Lodge [b]	57.40	$34.20 - $84.90	137,000	1,620,000
Bald Mountain	49.00	$29.00 - $68.00	≈2,600	Unknown
Conway Granite	59.30	$36.70 - $93.70	(c)	35,000,000

(a) includes estimates for probable and potential reserves
(b) average costs
(c) ample to meet maximum demand

Low cost resources that would be developed to satisfy near-term requirements for thorium oxide include the stockpiles in Fernald, Ohio; Miamisburg, Ohio; Curtis Bay, Maryland; Chattanooga, Tennessee and the thorite deposit at Hall Mountain. Approximately 8,000 short tons of thorium oxide are available from these resources at a cost below $10/lb. These resources could adequately supply thorium oxide for prototype purposes and small-scale commercialization of the thorium fuel cycle.

The longer-term economic thorium deposits include Wet Mountain, Colorado; Lemhi Pass, Idaho; and Palmer, Michigan. These deposits represent

approximately 160,000 short tons of thorium oxide. The resources could support thorium reactor-growth and development for at least the next 50 years under the most likely demand conditions with costs ranging from $8 to $17/lb.

Long term deposits for thorium include Bald Mountain, Wyoming; Bear Lodge, Wyoming; and Conway, New Hampshire. These deposits represent over 17 million tons of thorium oxide. The resources could support a thorium fuel cycle for hundreds of years. Costs approximately equal or exceed $50/lb of thorium oxide.

Costs illustrated on the supply curve do not include mine reclamation. Charges for reclamation may decrease the attractiveness of the large surface-deposits such as Conway Granite, Bear Lodge, and Palmer, Michigan. Reclamation was not analyzed in this report because we assumed that during the time frame of this study the pits would remain open. Thus, when economic conditions favor mining the low-grade ore bordering the pit, the ore can be easily accessed.

The supply curve will aid in the evaluation of the thorium fuel cycle. Feedstock costs, the other costs of the fuel cycle, and fuel exposures can be used to calculate energy costs. Thus, the economic viability of the thorium cycle can be analyzed.

17.2 COMPARISON WITH U.S.G.S. ESTIMATES

The United States Geological Survey (U.S.G.S.) recently reassessed several principal thorium resources in the United States.[1] Thorium districts discussed both in our study and the new U.S.G.S. report include Lemhi Pass, Idaho-Montana; Hall Mountain, Idaho; Wet Mountains, Colorado; Bear Lodge, Wyoming; and the Piedmont Placers of North and South Carolina. Most of the estimates generated by the U.S.G.S. report agree with the cost estimates obtained in our study. Deposit comparisons are discussed in the following paragraphs.

Both studies show close agreement for the Hall Mountain resource. According to our study, the Hall Mountain deposit contains approximately 5000 tons of reserves at an estimated $6.30/lb ThO_2. The U.S.G.S. study reports

4540 tons of ThO_2 reserves available at under $5/lb. The U.S.G.S. also esti-
mates 29,100 tons of probably-potential reserves which are beyond the scope of
our study.

Small discrepancies exist in the Wet Mountain resource analyses. The Wet
Mountains deposit contains 54,100 reserves available at under $15/lb according
to the U.S.G.S. and 9820 tons of reserves available at higher costs. Approxi-
mately 60,000 tons of ThO_2 reserves are reported in our study at $8.30/lb ThO_2.
The new and better data collected during the recent U.S.G.S. thorium reassess-
ment program has enabled them to more accurately determine reserve data. How-
ever, the discrepancies in the amount recoverable has a negligible effect on
the supply curve. Both studies illustrate the importance of the Wet Mountains
area in the closer-term supply picture.

The U.S.G.S. utilizes larger reserve estimates than our study. According to
the U.S.G.S. approximately 54,100 tons of reserves could be produced from
Lemhi Pass at costs under $15/lb of ThO_2, 2540 tons of reserves at costs
between $15/lb and $30/lb, and 7,280 tons of reserves at costs between $30/lb
and $50/lb. Our study evaluates 23,000 tons at $12/lb to $16/lb of ThO_2.
Part of the discrepancy can be resolved because the U.S.G.S. study assumes
higher ore grades and higher mine recoveries than we assumed for this analysis
resulting in recovery of more thorium per pound of ore. Also, additional
veins were discovered and included in Staatz reserve estimate based on data
unavailable during the course of our study.

Thus, there are probably additional reserves at Lemhi Pass not indicated
on our supply curve. We did not undertake additional analysis with the new
U.S.G.S. data because the additional reserve would cause only a minor change
in the overall supply curve.

The North and South Carolina placer deposits were also evaluated. Because
of their small size and low grade, both studies conclude that none of the
Piedmont placers could be operated at costs less than $50/lb ThO_2. This study
indicates the cost would exceed $100/lb ThO_2.

The Bear Lodge, Wyoming deposit was also discussed by the U.S.G.S., how-
ever cost analysis was not completed. An earlier report by Staatz[2] estimated

228,000 tons of ThO_2 potentially exist in the Bear Lodge Mountains. This study estimates it would cost approximately 57.00/lb ThO_2 to develop these reserves because of the difficulty in beneficiating the ore and concentrating the thorium.

Several resources discussed in our report but beyond the scope of the U.S.G.S. study are located in Palmer, Michigan, Bald Mountain, and Conway, New Hampshire. These deposits offer substantial quantities of thorium and play potentially important roles in the thorium supply picture.

Several deposits are analyzed in the U.S.G.S. study, but are not included in our study (see Section 5.0). Although many of these deposits may someday be developed for thorium, they were excluded from our study due to lack of data, their small size, or their lack of economic feasibility.

The supply curve for thorium illustrates the importance of various deposits in the domestic supply picture. As new exploration programs are undertaken, more thorium will probably be found in all ranges of production costs. Current data indicates, however, that there are abundant sources of thorium in the United States with economically feasible production costs.

REFERENCES

1. M. H. Staatz, T. J. Armbrustmacher, J. C. Olson et al., Principal Trorium Resources in the United States, U.S.G.S. circ. 805, U.S. Geological Survey, Co., 42 pp., 1978.

2. M. H. Staatz, Proceedings of the Uranium Industry Seminar, 1978, GJO-108(78), U.S. Dept. of Energy, Grand Junction, Colorado, pp. 109-115.

APPENDIX A

BUILDING AND EQUIPMENT COSTS

The following tables show the detailed building and equipment costs for each
thorium facility described in the text.

TABLE A-1.1. Stockpiled Thorium Buildings and Equipment Cost for Refinery

Equipment	Cost
2 - 100 gal Dissolvers	$ 10,000
2 - 100 gal Decantors	10,000
2 - 100 gal Adjustment Tanks	10,000
60 - Mixer Settlers	77,000
3 - 100 gal Boil Down Tanks	105,000
1 - Single Effect Evaporator	92,000
1 - Condensor	9,000
2 - 40 gal Mixers	11,000
1 - Rotary Vacuum Filter	37,000
1 - Vacuum Pump	5,000
1 - Plate and Frame Press	6,000
1 - Twin Screw Dryer	25,000
1 - Variable Speed Feeder	2,000
1 - Rotary Calciner	32,000
1 - Conveyor	2,000
1 - Elevator	2,000
1 - Drum Loader	4,000
1 - 400 gal Carbonate Mixing Tank	7,000
1 - 1000 gal HNO_3 Mixing Tank	11,000
1 - 70,000 gal HNO_3 Storage Tank	13,000
1 - 1000 gal Raffinate Holding Tank	5,000
2 - 2000 gal Solvent Storage Tanks	13,000
1 - 15,000 gal Water Tank	8,000
30 - Pumps (1" x 2")	30,000
Process Equipment Capital Cost (PECC)	$ 526,000
Field Erection (14% of PECC)	74,000
Foundations (12% of PECC)	63,000
Piping and Chuting (22% of PECC)	116,000
Electrical (19.5% of PECC)	102,000
Instrumentation (5% of PECC)	26,000
Miscellaneous Site Preparation (5% of PECC)	26,000
Buildings (35% of PECC)	185,000
Service Facilities (55% of PECC)	289,000
Land (6% of PECC)	31,000
Total Physical Cost (TPC)	$1,438,000
Engineering and Construction (25% of TPC)	359,300
Direct Plant Cost (DPC)	$1,797,000
Contractors Fee (5% of DPC)	90,000
Total Plant Cost	$1,887,000
Land	-31,000
TOTAL Building and Equipment Costs	$1,856,000

TABLE A-1.2. Stockpiles Effluent Control Buildings and Equipment Cost
for Refinery

6000 cfm Bag House Filter	10,700
6000 cfm HEPA Filter	21,600
Dehumidifier/Heater Unit	2,200
Process Equipment Capital Cost	34,500
Field Erection (14% of PECC)	4,800
Foundations (12% of PECC)	4,100
Piping and Chuting (22% of PECC)	7,600
Electrical (19.5% PECC)	6,700
Instrumentation (5% of PECC)	1,700
Miscellaneous Site Preparation (5% of PECC)	1,700
Buildings (35% of PECC)	12,100
Service Facilities (55% of PECC)	19,000
Land (6% of PECC)	2,100
Total Physical Cost	94,300
Engineering and Construction (25% of TPC)	23,600
Direct Plant Cost	117,900
Contractor's Fee (5% of DPC)	5,900
Total Plant Cost	123,800
Land	-2,100
Total Buildings and Equipment Cost	121,700

TABLE A-2.1. Lemhi Pass Buildings and Equipment Cost for Mill

	Component	Total
1-	Run of Mine Storage	30,000
2-	Front End Loader	120,000
1-	Primary Crusher	55,000
1-	Secondary Crusher	97,000
2-	Vibrating Screens	50,000
1-	Sampler	20,000
1-	Crushed Ore Bin	19,000
2-	Ball Mills	620,000
2-	2100 Gal. Agitated Tank	70,000
2-	2500 Gal. Agitated Tank	74,000
12-	3500 Gal. Agitated Tank	80,000
2-	4300 Gal. Agitated Tank	104,000
1-	Thickener	130,000
9-	Pressure Leaf Filter	162,000
2-	Vacuum Pump	60,000
7-	2500 Gal. Mixer/Settler	266,000
2-	Strip Solution Prep Tanks	20,000
1-	Disk Centrifuge	120,000
1-	Rotary Vacuum Filter	543,000
1-	Twin Screw Dryer	134,000
1-	Rotary Calciner	185,000
1-	Cyclone Dust Collector	12,000
1-	Bag Filter	10,000
1-	Centrifugal Fan	2,000
1-	Drum Loader	7,000
1-	550,000 Gal. Acid Tank	59,000
3-	Solvent Tanks	28,000
3-	Flocculant Tanks	20,000
1-	1,000,000 Water Tank	164,000
1-	Steam Generator	116,000
	Materials Handling Equipment	242,000
	Pumps	215,000
	Process Equipment Capital Cost	3,834,000

Field Erection (14% of PECC)	537,000
Foundations (12% of PECC)	460,000
Piping and Chuting (22% of PECC)	843,000
Electrical (19.5% of PECC)	748,000
Instrumentation (5% of PECC)	192,000
Misc. Site Preparation (5% of PECC)	192,000
Buildings (35% of PECC)	134,000
Service Facilities (55% of PECC)	211,000
Land (6% of PECC)	230,000
Total Physical Cost (TPC)	7,381,000
Engineering and Construction Overhead (25% of TPC)	1,845,000
Direct Plant Cost (DPC)	9,226,000
Contractors Fee (5% of DPC)	461,000
Total Plant Cost	9,687,000
Land	230,000
Total Buildings and Equipment Cost	9,917,000

TABLE A-2.2. Lemhi Pass Effluent Control Buildings and Equipment Cost
for Mill

26,000 cfm Cyclone	$ 6,600
21,000 cfm Wet Impingement Scrubber	$ 13,300
4,000 cfm Baghouse Filter	$ 7,600
Process Equipment Capital Cost	$ 27,500
Field Erection (14% of PECC)	3,800
Foundations (12% of PECC)	3,300
Piping and Chuting (22% of PECC)	6,000
Electrical (19.5% of PECC)	5,400
Instrumentation (5% of PECC)	1,400
Miscellaneous Site Preparation (5% of PECC)	1,400
Buildings (35% of PECC)	9,600
Service Facilities (55% of PECC)	15,000
Land (6% of PECC)	1,600
Total Physical Cost	$ 75,000
Engineering and Construction (Overhead (25% of TPC)	18,700
Direct Plant Cost	93,700
Contractor's Fee (5% of DPC)	4,700
	98,400
Land	1,600
	$ 96,800
Water Tank Truck	30,000
Storage Pile Barriers	15,000
	$142,000

TABLE A-2.3. Lemhi Pass: Sulfuric Acid Plant Buildings and Equipment Cost

Summary of Predesign Capital Cost Estimate for
Thorite Mill Sulfuric Acid Plant

Plant Total Physical Cost	1,818,000
Engineering and Construction Overheads (25% of TPC)	454,000
Direct Plant Cost	2,272,000
Contractors Fee (5% of DPC)	114,000
Total Plant Cost	2,386,000
Land	109,000
	2,495,000

TABLE A-3.1. Palmer, Michigan Buildings and Equipment Cost for Refinery

Equipment	Cost
1 - Monazite Storage Hopper	$ 4,700
3 - Ball Mill	180,000
10 - 600 gal. Agitated Tanks	552,000
1 - Leaf Type Pressure Filter	223,800
9 - 600 gal. Mixer Settler	337,900
1 - Acid Mixer	21,000
1 - Strip Solution Mixer	21,000
1 - Caustic Mixer	21,000
1 - 240 sq. ft. Drum Filter	112,600
1 - 150 sq. ft. Drum Filter	87,500
1 - 3000 sq. ft. Screw Dryer	119,000
1 - 6' x 50' Rotary Calciner	450,000
1 - 2000 CFM Fan	2,100
1 - 6000 CFM Cyclone Dust Collector	10,600
1 - 6000 CFM Bag Filter	12,000
1 - Bag Machine	9,800
2 - Vacuum Pumps	92,800
Material Handling Equipment	110,000
Pumps	215,000
Process Equipment Capital Cost	$2,582,800
Field Erection (14% of PECC)[1]	361,600
Foundations (12% of PECC)[1]	309,900
Piping and Chuting (22% of PECC)[1]	568,200
Electrical (19.5% of PECC)[1]	503,600
Instrumentation (5% of PECC)[1]	129,100
Miscellaneous Site Preparation (5% of PECC)[1]	129,100
Building (35% of PECC)[1]	904,000
Service Facilities (55% of PECC)[1]	1,420,500
Land (6% of PECC)[1]	155,000
Total Physical Cost (TPC)	$7,063,800
Engineering and Construction (25% of TPC)[1]	1,765,900
Direct Physical Cost (DPC)	$8,829,700
Contractor's Fee (5% of DPC)[1]	441,500
Total Plant Cost	$9,271,200
Land	-155,000
	$9,116,200

TABLE A-3.2. Palmer, Michigan Effluent Control Buildings and Equipment
for Refinery

Equipment	Cost
1 - 10,000 CFM Baghouse Filter	$ 16,500
1 - 16,000 CFM Venturi Scrubber	12,300
1 - 3,000 CFM Baghouse Filter	5,900
Total	$ 34,700
Field Erection (14% of PECC)	4,900
Foundations (12% of PECC)	4,200
Piping and Chuting (22% of PECC)	7,600
Electrical (19.5% of PECC)	6,800
Instrumentation (5% of PECC)	1,700
Miscellaneous Site Preparation (5% of PECC)	1,700
Buildings (35% of PECC)	12,100
Service Facilities (55% of PECC)	19,100
Land (6% of PECC)	2,100
Total Physical Cost	$ 94,900
Engineering and Construction (25% of TPC)	23,700
Direct Plant Cost	$118,600
Contractor's Fee	29,600
Total Plant Cost	$148,200
Land	-2,100
Total Buildings and Equipment Cost	$146,100

TABLE A-3.3. Palmer, Michigan Sulfuric Acid Plant Buildings and Equipment
Cost

Equipment	Total Cost
Plant Total Physical Cost[3]	$3,195,000
Engineering and Construction (25% of TPC)[1]	436,700
Direct Plant Cost	$3,993,750
Contractor's Fee (5% of DPC)[1]	109,200
Total Plant Cost	$4,193,437
Land	-70,091
Total Buildings and Equipment	$4,123,000

TABLE A-3.4. Palmer, Michigan: Byproduct Recovery Buildings
and Equipment for Refinery

Equipment	Total Cost
2 - 30,000 gal. Agitated Tanks	$ 276,000
1 - 3,000 sq. ft. Rotary Filter	320,000
2 - 20,000 gal. Agitated Tanks	202,400
1 - 1,500 sq. ft. Rotary Filter	200,000
1 - 500 sq. ft. Rotary Dryer	110,000
1 - Bag Packer	80,000
1 - 24,000 CFM Wet impingement Scrubber	14,900
Pumps	50,000
Materials Handling	120,000
Process Equipment Capital Cost	$1,373,300
Field Erection (14% of PECC)	192,300
Foundation (12% of PECC)	164,800
Piping and Chuting (22% of PECC)	302,100
Electrical (19.5% of PECC)	267,800
Instrumentation (5% of PECC)	68,700
Miscellaneous Site Preparation (5% of PECC)	68,700
Buildings (35% of PECC)	480,600
Service Facilities (55% of PECC)	755,300
Land (6% of PECC)	82,400
Total Physical Cost	$3,756,000
Engineering and Construction (25% of TPC)	939,000
Direct Plant Cost	$4,695,000
Contractor's Fee (5% of DPC)	234,700
Total Plant Cost	$4,929,700
Land	-82,400
	$4,847,000

APPENDIX B

FIXED CHARGE RATE

APPENDIX B

CALCULATION OF FIXED CHARGE RATES

i_e = Weighted average cost of equity as a fraction (preferred stock and common equity).

i_b = Bond interest rate as a fraction.

b = Fraction of capital provided by bonds.

k = State tax (1 - Fed Tax) + Fed Tax

n = Economic life of facility in years.

v = Property tax rate.

i = Effective cost of capital to the utility taking into consideration the tax credit for bond interest = $b \cdot i_b(1-k) + (1-b)i_e$.

crf = Capital recovery factor (level annual fraction of initial capital required to recover investment plus interest) = $\dfrac{i(1+i)^n}{(1+i)n-1}$.

pwf = Uniform series present worth factor = 1/crf.

tcf = Tax credit factor (level annual fraction for depreciation credit equivalent to a sum-of-years digits depreciation schedule).

$$= \frac{2}{N(n+1)i} \left[\frac{n}{pwf} - 1\right]$$

fc = Fixed charges to recover investment, pay taxes, pay bond interest and return on equity

$$= crf + \frac{k}{(1-K)} \left[crf - tcf\right]$$

tax = Level annual income taxes

$$= k(fc - b \cdot i_b - tcf).$$

sf = Sinking fund (level annual fraction of capital which, when reinvested at rate i, produces the original investment after n years)

$$= \frac{1}{(1+i)^n - 1}$$

p.tax = Level annual property tax equivalent to tax on depreciated investment using straight line depreciation

$$= V \left[1 - \frac{1}{n} \left(\frac{1}{i} - \frac{n}{(1+i)^n - 1} \right) \right]$$

Total fixed charges are the sum of the following items:

Interest on Bonds ($i_b \cdot b$)

Return on Equity $\left[i_c \cdot (1-b) \right]$

Recovery of Capital (sf)

Federal and State Income Taxes (tax)

Property Tax (p.tax)

Property Insurance Allowance

Interim Replacements Allowance

State Revenue Tax (Sum of above ÷ (1 - Revenue Tax Rate))

APPENDIX C

EFFECT OF URANIUM DECAY DAUGHTER
(Th-230) ON THORIUM RECOVERY

SUMMARY

In ores containing both thorium and uranium, the economic effect of a decay product of uranium, Th-230, has been questioned in thorium-232 reactor fuel. U-232 is formed by irradiation of Th-230 and decays into daughter products emitting penetrating gamma radiation. Thus, U-233 fuel containing U-232 in great enough concentration must be remotely fabricated. Utilizing thorium containing essentially zero Th-230 has been hypothesized as a means of controlling U-232. However, studies accomplished using the ORIGEN[1] code indicate that n, 2n reactions of neutrons with Th-232 also produce significant quantities of U-232. Thus, remote fabrications of U-233 will be necessary regardless of the Th-230 content of the ThO_2 reactor fuel unless current reactor design or operating procedure is modified. Since the effectiveness of these modifications is questionable, no extra economic considerations are warranted in regard to Th-230 level for ores in this study.

INTRODUCTION

Thorium is a valuable source of fertile material. Thorium by itself is not fissionable, but when natural thorium (Th-232) is irradiated, the fissile isotope U-233 is formed. U-233 is then separated from the thorium and fabricated into nuclear fuel. The thorium is either regenerated and returned to the reactor for further irradiation or discarded.

Fabrication of the U-233 may have to be conducted remotely because of the presence of the isotope U-232 which has daughter products emitting high energy gamma rays. U-232 is mainly attributed to the presence of thorium-230 in the original thorium-232 fuel. Th-230, in turn, is derived from the decay of uranium contained in some thorium-bearing ores.

The purpose of this report is to determine whether the presence of Th-230 in natural thorium will warrant special economic considerations. The report discusses: (1) when thorium-230 will be present in thorium reactor fuel, (2) the relationship between Th-230 and U-232, and (3) the importance of Th-230 in the formation of U-232 inside the reactor. The above information will then be related to the ores in this study and also compared to ores containing large quantities of Th-230. Finally, possible alternatives which affect the results of this study will be discussed.

Occurrence of Th-230

Naturally occurring Thorium-230 is present in thorium ores associated with uranium. Uranium decays into Thorium-230 by the following reaction:

$$\text{U-238} \xrightarrow[4.5 \times 10^9 y]{\alpha} \text{Th-234} \xrightarrow[24d]{\beta} \text{Pa-234} \xrightarrow[6.7h]{\beta} \text{U-234} \xrightarrow[2.5 \times 10^5 y]{\alpha} \text{Th-230}$$

Thorium will thus be present in natural thorium (Th-232) uranium ore mixtures at some equilibrium concentration. Since Th-232 and Th-230 are chemical isomers, they cannot be chemically separated and the refined ThO_2 will be a mixture of both isotopes.

Th-230 concentrations in Th-232 for ores in this study can be computed assuming the uranium equilibrium concentration of 18 ug Th-230/gU[2]. The thorium to uranium ratio of the ore is then estimated for different thorium deposits. Table C.1 illustrates the various concentrations of Th-230 in Th-232 for several thorium-containing ores. The Th-230 concentration varies from negligible to 60 ppm.

Relationship Between Th-230 and U-232

When thorium oxide containing Th-230 is irradiated, U-233 and U-232 are formed by the following reactions:

$$\text{Th-232} \xrightarrow{n,\gamma} \text{Th-233} \xrightarrow[23 \text{ min.}]{\beta} \text{Pa-233} \xrightarrow[27.4d]{\beta} \text{U-233}$$

$$\text{Th-230} \xrightarrow{n,\gamma} \text{Th-231} \xrightarrow[25.5h]{\beta} \text{Pa-231} \xrightarrow{n,\gamma} \text{Pa-232} \xrightarrow[1.31d]{\beta} \text{U-232}$$

U-233 is a long-life fissile isotope of uranium and utilized as a reactor fuel. U-232 is a useless activation product which in itself is not detrimental. However, U-232 decays into a series of short-lived daughter products, some of which emit penetrating gamma radiation. Thus, after U-233 and U-232 are chemically separated from irradiated thorium, the U-232 decay products gradually build up in gamma activity. As a result, fabrication either has to be carried out quickly or will have to be conducted remotely[3].

TABLE C.1. Th-230 Content in Domestic Ores

Source	$\dfrac{\text{Th}}{\text{U}}$	$\dfrac{\text{ug Th-230}}{\text{g Th-232}}$
Thorite (Lemhi Pass)	$\dfrac{100^b}{1}$	.17[a]
Granite	$\dfrac{15^c}{4}$	4.8
Placer Monazite	$\dfrac{23^d}{1}$	.77
Conglomerate Monazite	$\dfrac{23^e}{1}$	.77

(a) Analysis by C. A. Rohrman from personal communication between J. E. Minor, and E. R. Astley, Exxon Nuclear, Feb. 17, 1976.
(b) Francis J. Kelly, Technological and Economic Problems of Rare-Earth Metal and Thorium Resources in Colorado, New Mexico, and Wyoming. Information circular 8124, United States Department of Interior-Bureau of Mines, Update 1976, p. 18.
(c) Nishimori, Richard K., et al., Uranium Deposits in Granitic Rocks, A report to the Energy Research and Development Administration, January, 1977, p. 7.
(d) Cutbert, Thorium Production Technology, Addison-Wesley Publ. Co., Inc., Reading, Mass., 1958, p. 53.
(e) S. R. Borrowman and J. B. Rosenbaum, Recovery of Thorium from A Wyoming Ore, USBM Inf. report 5916, 1961.

Importance of Th-230 in U-232 Formation

As shown in the previous paragraph, irradiation of Th-230 results in the formation of U-232, but U-232 is also formed in reactor fuels as follows:

1. Th-232 $\xrightarrow{n,2n}$ Th-231 $\xrightarrow[25.5h]{\beta}$ Pa-231 $\xrightarrow{n,\gamma}$ Pa-232

$\qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad$ 1.32d $\downarrow$ β

$\qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad$ U-232

2. U-233 $\xrightarrow{n,2n}$ U-232

3. U-235 $\xrightarrow{n,\gamma}$ U-236 $\xrightarrow{n,\gamma}$ U-237 $\xrightarrow[6.75d]{\beta}$ Np-237

$\qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad\qquad$ $\downarrow$ n, 2n

U-232 $\xleftarrow[2.85Y]{\alpha}$ Pu-236 $\xleftarrow[22h]{\beta}$ Np-236

Assuming the above reactions form only negligible U-232, utilization of thorium with low Th-230 content will limit the formation of U-232. Thus, the higher costs associated with remote fabrication could be avoided by using fuel with a negligible Th-230 content.

To estimate the importance of Th-230 in the formation of U-232, a sensitivity analysis was conducted using the ORIGEN computer code.[1] The calculation is based on a typical LWR cell of UO_2 fuel design, a goal exposure of 33,000 MWD/t, and one year of post irradiation cooling. The results are shown as follows:

Recoverable ^{232}U for various ^{230}Th concentrations

ppm ^{230}Th in ^{232}Th	Grams per ton of Uranium Enriched
1000	133
100	30.6
10	20.5
0	19.3
0*	0.234

---

*Th^{232} cross section for the n, 2n reaction set to zero.

The results indicate that even with no Th-230 in the reactor fuel, 19.3 g U-232 will be present in recoverable uranium per ton of thoria fuel. Assuming approximately 34,500 grams of recoverable uranium per ton initial heavy metal, approximately 560 ppm of U-232 will be present in the discharged uranium. Multiplying this by 3.84 mR/hr/ppm[4] gives a dose rate of 2.15 R/hr. An acceptable dose rate for restricted personnel working 40 hrs/week is 2.9 mR/hr according to 10 CFR 20. Thus, remote fabrication will be required for even very low Th-230 content of the original thorium. In addition, the U-232 resulting from a hypothetical fuel containing Th-232 with a zero n, 2n cross-section is almost negligible. Therefore, much of the U-232 in recoverable U-233 results from (n, 2n) reactions on the Th-232.

Thus, it appears that for current thermal reactor designs, remote fabrication will be necessary and no premium is warranted for ore with low Th-230 content.

Effect of High Th-230 Concentrations

For ores in this study, Th-230 content in Th-232 ranges from almost zero to approximately 5 ppm (Table C.1). According to Battelle's analysis, the thorium-230 content of these ores warrants little concern. Even for ores containing 1000 ppm Th-230 in Th-232, only a 7-fold increase in U-232 content in recoverable uranium is realized. According to data calculated at Battelle, a ten-fold increase in dose will require only an additional 1 ft. of concrete shielding.

Extremely high Th-230 concentrations must be avoided, however. In the case of residues from typical uranium ore mills, Th^{230} in total thorium range from .08-3.7%[1]. Th-230 ranges of this nature would not only pose increased shielding requirements, but the dilution of Th-232 would result in lower reactor conversion ratios. As mentioned previously, ores in this study contain significantly lower Th-230 in Th-232 concentrations.

Possible Uses of Th-230 Free Fuel

Modifications to current fuel cell design have been proposed which limit n, 2n reactions with Th-232 and the formation of Th-230. An example of a proposed method is as follows: The thorium charges can be loaded in zones separate from the fissile fuel zone. This, in turn, reduces the amount of fast neutron flux in the thorium zone. Since fast neutrons (6 MeV) result in n, 2n reactions with Th-232, reducing the fast neutron flux reduces the formation of U-232. Unfortunately, the thermal neutron flux is also decreased and consequently a lower U-233 conversion ratio results.

The goal exposure of the thoria fuel can also be reduced. Currently, a burn-up of 30,000 MWD/ton for an LWR cell can be assumed. If we reduce this exposure by one-third to 10,000 MWD/ton, approximately one-third the U-233 will be bred inside the reactor. However, the formation of U-232 is dependent on two neutron captures and varies exponentially with time to approximately the second power. Thus, reducing the exposure by one-third will reduce U-232 formation by one-ninth.

If these methods prove to be adequate, Th-230-free fuel might be advantageous. However, the economic considerations of higher reactor operation costs vs. higher fabrication costs would have to be analyzed and is beyond the scope of this report.

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C.9

APPENDIX D

LEMHI PASS MODEL MINE

This appendix is a report completed for PNL by a consulting agency.

Golder Associates

CONSULTING MINING AND GEOTECHNICAL ENGINEERS

REPORT TO
BATTELLE–PACIFIC NORTHWEST LABORATORIES
FOR
MINING THORIUM–BEARING VEINS
AT LEMHI PASS
BY UNDERGROUND MINING METHODS

Distribution:

      3 copies – Battelle-Pacific Northwest Laboratories

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S78008   June 1978

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1.0 INTRODUCTION

This report complies with the request of Battelle-Northwest
to investigate the cost of mining thorium-bearing veins by
underground mining methods at Lemhi Pass, Idaho, as described in
Consultant Agreement B-65327-A-L, Prime Contract EY-76-C-06-
1830. It forms part of a thorium resource price analysis to
provide the data needed for the fuel cycle economic studies
under investigation by Savannah River Laboratory and Pacific
Northwest Laboratories.

Thus, to furnish the necessary data, conceptual mining
approaches for the underground working of the thorium deposits
were postulated in conjunction with an estimate of their related
capital and operating costs.

These estimates are based on an appropriate orebody model
and various production parameters hypothesized from the limited
geological information available.

2.0 SUMMARY

2.1 Description of the Lemhi Pass Area

The Lemhi Pass area is situated in the east-central part of the Beaverhead Mountains along the Idaho-Montana border in the northwestern United States.

For the most part, the area is uninhabited, and the Beaverhead Mountains are a generally rugged, single-crested divide that trends southeast and separates the valley of the Lemhi River on the west from the Horse Prairie of the Beaverhead Valley on the east. Drainage has dissected the slopes of the range into a series of deep gorges and steep ridges that are heavily timbered on the north-facing slopes and also in shaded canyons. The district lies within the Salmon National Forest and will have to comply with the new environmental regulations.

2.2 Geological Information

The level of geological data available upon which to base an orebody model is low, and therefore any models developed must be regarded as hypothetical.

The major groups of rocks exposed in the Lemhi Pass area are fine-grained clastic sedimentary rocks (Precambrian Belt Supergroup), volcanic rocks (Tertiary Challis Volcanics) and unconsolidated deposits (Quaternary). These rocks have been folded and faulted and have been considerably disturbed a number of times. A generally low competence of the country rocks is indicated.

2.3 Hypothetical Orebody Model

The thorium-bearing veins are irregular siliceous replacement bodies formed along zones of shearing and faulting. The vein geometry is irregular but may be regarded as a tabular narrow body with a steep dip of about 60 degrees and a strike length from 50 feet to approximately 1,000 feet. Vein thicknesses are usually less than 5 feet but can be 10 feet and greater. Mineralization is also extremely variable with an overall mean for the district of 0.46 percent thorium.

2.4 Conceptual Mining Methods

Given the narrow veins, which dip steeply in rocks of low competence, two stoping methods are postulated; namely, the resue, and the cut and fill methods. Veins narrower than 5 feet are mined by resuing and veins of greater thickness by cut and fill methods.

2.5 Project Costs

The total cost of mining the thorium deposits at a production rate of 250,000 tons per year is $71,389,734. Operating costs account for $63,781,234 of this sum and capital expenditure amounts to $7.608,500. The upper and lower bounds for the total project cost are $105,000,000 and $64,400,000 respectively.

2.6 Major Uncertainties

The major areas of uncertainty in this report which could affect the costs estimated, include geological data such as vein geometry and ore grade, ground conditions and the cost of stope fill.

3.0 DESCRIPTION OF THE LEMHI PASS AREA

The Lemhi Pass area lies astride the Continental Divide along the Idaho-Montana border in the northwestern United States (see Figure 1). This area represents a zone of abundant thorium-bearing veins.

3.1 Location

The major Part of the Lemhi Pass area is in Beaverhead County, Montana, and is sited approximately 40 miles southwest of Dillon, Montana. The remaining area is in Lemhi County, Idaho, and is some 25 miles southeast of Salmon, Idaho. The district lies between State Highway 28 to the west and Interstate Highway 15 to the east. An unimproved dirt road that crosses the northern part of the Lemhi Pass quadrangle connects these major highways. Roads and trails lead off from this county road and give access to most of the properties and old mine sites in the area. However, travel over some sections of these roads requires the use of four-wheel drive vehicles. To the east of the area a line of the Union Pacific Railroad runs along the east side of the Clark Canyon Reservoir, Montana.

Although, for the most part, the area is uninhabited, it is envisaged that future mining operations could be staffed from the populations of the outlying towns such as Salmon and Dillon.

3.2 Geography

The Beaverhead Mountains in this area are a generally rugged, single-crest divide that trends southeast and separates the valley of the Lemhi River on the west from the Horse Prairie of the Beaverhead Valley on the east. The range rises steeply from the flanking valleys to reach elevations in places of almost

11,000 feet but at the Lemhi Pass an altitude of only 7,373 feet
is attained. In cross-section, the range is asymmetric with a
generally irregular eastward slope that is less abrupt than the
more unbroken front of the western slope (Figure 2). Thus, due
east of Lemhi Pass the country is characterized by rolling
barren hills. Glacial cirques are prominent on the east side of
the crest of the divide which is broadly rounded and is almost
park-like for several miles in the area of the broad shallow
saddle of Lemhi Pass. However, the crest becomes more rugged as
well as higher in altitude both to the south and north of the
Pass area.

Drainage has dissected the slopes of the range into a
series of deep gorges and steep ridges that are heavily timbered
on the north-facing slopes and also in shaded canyons. The
general pattern of both open and forested areas is controlled by
the amount of moisture obtained and held by the surficial soil,
as well as by the rock type. In general, the area underlain by
volcanic rocks is less forested than that underlain by other
rocks. For example, the roughly triangular patch of volcanic
rocks near the center of the area is for the most part
unforested but is surrounded by forest-covered areas that are
underlain by Belt rocks concealed by glacial till. Yellow pine,
lodgepole pine, douglas fir, and aspen are the principal trees
of the area; limber pines grow at places on the highest crest.
Unforested areas are heavily covered with grasses and sparse
amounts of sagebrush.

3.3 Climate

Snow is abundant from mid-October to mid-May with average
accumulations of approximately 36 inches. During these fall,
winter, and spring months the air temperature can drop to -40°F
but is more usually in the region of 45°F. The summers are mild

EXPLANATION

QUATERNARY
Alluvium
Landslide Deposits
Terrace Deposits
Younger Glacial Deposits
Older Glacial Deposits

TERTIARY
Diorite
Younger Basalt
Tuff of Curtis Ranch
Tuff of Lemhi Pass
Basalt myodacite sequence
Vitric Tuff of Lemhi Pass
Conglomerate of Flume Creek
Quartzite-bearing rhyolite tuff
Basal conglomerate

PRECAMBRIAN
Belt Supergroup

Quartz Latite

Vein
Fault
Proposed Underground Haulage

LEMHI PASS FAULT
DAN PATCH FAULT
BULL MOOSE FAULT
TERRAN CREEK FAULT

Continental Divide
SALMON NATIONAL FOREST
BEAVERHEAD NATIONAL FOREST
LEMHI PASS
IDAHO
MONTANA

COPPER QUEEN VEIN
SOUTH FORK AGENCY CREEK
LUCKY STRIKE VEIN
TREASURE No. 2 VEIN
LAST CHANCE VEIN
SHEAR ZONE VEIN
BROWN BEAR VEIN
FRYING PAN VEIN

A A'
B B'

Continental Divide
LEMHI PASS FAULT
BULL MOOSE FAULT
DAN PATCH FAULT
SHEAR ZONE VEIN
LAST CHANCE VEIN
DIORITE DIKE
TERRAN CREEK FAULT

D.7

and occasional showers and thunderstorms provide sufficient moisture to maintain the flow of all the creeks and numerous springs. The average annual rainfall is about 18 inches.

3.4 Environmental Considerations

Future mining operations in the Lemhi Pass district, which lies within the Salmon National Forest, will have to comply with the new regulations that fall under the "Mineral Resources on National Forests Use Under U.S. Mining Laws, Title 36, Code of Federal Regulations, Part 252." However, as mining operations have been carried out previously in this area, the revival of operational activities will probably be considered as a minimal disturbance of the area. Thus it is unlikely that environmental considerations will be of major importance to the overall project cost.

4.0 GEOLOGICAL INFORMATION

Previous geological work in the Lemhi Pass area dates back
to 1804 when an expedition was commenced by Lewis and
Clark (Sharp and Cavender, 1962). Exploration work has been
sporadic since the initial investigation, apart from two periods
of prospecting during which the geology of the area was further
studied. Copper was the subject of the first period which
started about 1883 and resulted in the discovery of the main
vein in the Copper Queen, a small underground mine. The second
period was initiated with the finding of thorium in the Lemhi
Pass area in 1949. Prospecting activity has been maintained
throughout this latter period with the emphasis on the search
for radioactive minerals.

The greater part of the work on the thorium veins
concentrated on their size, shape, and grade. These data were
obtained by shallow bulldozer trenches, short adits (with the
exception of the Last Chance vein), drifts, and three
boreholes (Sharp and Cavender, 1962). Recovery from these
boreholes was on the whole poor, being 51 percent (DDH1), 39
percent (DDH3), and 0 percent (DDH4). Further, as the
overburden is thick in many areas and outcrops are few, the
position of contacts of the veins had to be inferred in many
places. From this level of information it can be seen that any
orebody model developed must at best only be regarded in a
hypothetical light.

Of the geological studies (see references, this report)
which have been carried out in the Lemhi Pass area over these
years, the most recent work of Sharp and Cavender (1962) and
Staatz (1972) is considered to be the most appropriate to this
report. The former used aerial mapping and detailed mapping by
plane table methods for the surface exposures and Brunton
compass and tape for the underground mapping. In addition to

some of these techniques for mapping the regional geology, Staatz (1972) in his detailed study of the mineralogy of the thorium veins, separated individual minerals by heavy liquids, magnetic separation, and hand picking. Minerals were then identified by X-ray diffraction. All the known underground workings were also investigated by Staatz for radioactivity with a scintillation counter.

4.1 Geological Setting

The major groups of rocks exposed in the Lemhi Pass area are fine grained clastic sedimentary rocks (Precambrian Belt Supergroup), volcanic rocks (Tertiary Challis Volcanics) and unconsolidated deposits (Quaternary), (Figure 2). The Precambrian Belt Supergroup consists principally of fine-grained micaceous quartzite, interbedded in places with siltite. These rocks underlie the entire district and are poorly exposed at the surface over about 60 percent of the area. With one exception, all veins in the Lemhi Pass area occur in rocks of the Belt Supergroup which probably at one time was entirely overlain by flows and tuffs of the Tertiary Challis Volcanics. These volcanic rocks were subsequently eroded and are now exposed in only about 15 percent of the area in downfaulted blocks and small erosional remnants (Figure 2). The consolidated Quaternary deposits that conceal the underlying rocks in about 25 percent of the area consist of older glacial deposits, river gravels, younger glacial deposits, terrace deposits, landslide deposits and alluvium.

Diorite dikes, of post-Challis age, are scattered throughout the area.

4.2 Structure

The Precambrian rocks in the Lemhi Pass area were folded
and faulted by the forces that produced the Beaverhead Mountains
and other ranges of east-central Idaho and southwest Montana.
Tertiary rocks of the district are also folded and faulted but
to a much lesser degree. So far as is known, the folds are
small and the major structural features in the area are long and
continuous forming wide broken-rock zones, but most of the
faults appear to be relatively short and only a few feet wide
(Figure 2). Many such faults that are present along the
Continental Divide south of the pass trend principally north to
northwest, but other faults in the same area have diverse
trends. The faults all have steep to near-vertical dips.

The beds of the Belt series are generally thin, from 1
inch to a few feet, and mostly dip from about 20 to 50 degrees
to the north. Minor folds are present, and more complex folding
is suggested by some of the mapping data. A conspicuous
fracture cleavage is present in the same beds which dips at a
much lower angle than the Belt rocks.

The most prominent structural feature in the area is the
zone of faulting called the Lemhi Pass fault which forms a zone
350 to 1,000 feet wide. The dip of these faults is, in general,
to the south at 60 degrees or more.

In places, the bedded tuffs are inclined as much as 35
degrees which indicates considerable post-volcanic tilting in
the area. The Tertiary lake beds along the flanks of the
Beaverhead range in the Lemhi Pass area are generally inclined
20 to 25 degrees to the east.

4.3 Mineral Deposits

The greater part of the mineralization in the area occurs
in quartz veins and irregular siliceous replacement bodies that
contain copper, gold, and thorium. Veins were noted by Staatz
(1972) at 107 localities and, of these, 87 were found to contain
thorium. Base and precious metals are present in some of the
thorium deposits and in other veins of the district and along
its border. However, only the Copper Queen mine and the
Blue Bird mine near Agency Creek in Idaho have actually produced
copper. The area is also known to contain several thousand tons
of rare earths.

The veins were formed by fissure-filling and wall-rock
replacement along zones of shearing and faulting in the
micaceous quartzite of the Belt series. Recurrent movement
along the faults has broken and sheared most of the vein
material. The grade can therefore be expected to vary consider-
ably within the veins. For example, the 14 samples taken from
the Last Chance vein ranged from 0.074 percent to 1.7 percent,
with a mean of 0.45 percent and a coefficient of variation of
106 percent. It is therefore evident that the many single
samples recorded by Staatz have very little meaning other than
to confirm the presence of thorium.

4.3.1 Dimensions and Attitudes of Veins

Throughout the latest period of exploration the dimensions
and attitudes of the outcrops of the numerous veins in the area
have become better defined. For example, the length of the Last
Chance vein exposed in 1952 was about 1,200 feet in comparison
to its known length at the present time of approximately 3,900
feet. In order to produce a realistic model of the veins in the

district for the purposes of developing conceptual mining
methods, several of the veins have been studied. Dimensions of
these veins, as detailed by Staatz, are given in Table 1.

TABLE 1

DIMENSIONS AND ATTITUDES OF SOME OF THE

THORIUM-BEARING VEINS IN THE LEMHI PASS AREA

Vein	Exposed Length (feet)	Thickness (feet)	Strike	Dip
Lucky Horseshoe	1,400	0.8 - 1.5	N67°W	70°NE
Betty Jo	425	1.0	N35°E	20°NW
Buffalo	250	2.0 - 11.0	N5°-27°W	60°SW
Deer Fraction	60	2.0 - 3.0	N72°W	70°SW
Deer	250	2.0	N67°W	*
Wonder	1,200	1.0 - 10.0	E-W	40°-89°S
Cago No. 12	2,800	0.3 - 13	N55°W	46°-48°S
Cago No. 10	100	1.3	N45°W	Steep W
Contact	1,950	0.7 - 10.0	N70°E	26°-56°S
East Vein	60	+3.0	N39°W	Steeply
West Vein	50	4.5	N30°E	40°NW SW
Beaverhead	525	2.0 - 4.0	N85°E	40°S
Trapper No. 1	235	5.0 - 11.0	N75°-85°N	45°-60°S
Wonder East Vein	100	1.5 - 2.0	N38°W	30°SW W
Little Dandy No. 1	90	2.5	N75°W	80°NE
Shear Zone	6,000	0.5 - 2.5	N55°W	65°-81°SW
Black Rock	1,630	2.0 - 6.0	N47°W	70°W
Last Chance	3,900	7.0 - 30.0	N65°-80°W	43°-70°SW
Frying Pan West	100	3.0	N30°W	54°SW
Frying Pan Southeast	1,100	0.5 - 0.7	N36°-73°W	72°-78°SW
G & G No. 6 & 8	1,300	1.0 - 2.0	N48°W	Steep
G & G No. 1 & 2	1,600	0.8 - 2.0	N48°W	58°SW
Thorite No. 1	300	0.9	N45°W	85°NE
Dan Patch	1,050	0.3 - 2.5	N77°W	60°-80°SW
Reactor	1,650	0.3 - 3.0	N42°W	72°SW-78°NE

* Not determined.

It is evident from Table 1 that the thorium-bearing veins are irregular tabular bodies ranging in thickness from less than 1 foot to 30 feet. About 50 percent of the known veins average more than 1 foot thick and about 12 percent are more than 5 feet thick. The thickest vein is the Last Chance which is from 7 to 30 feet thick. Several other veins are as much as 10 feet thick. Most veins are extremely variable in thickness, but a few are fairly uniform. For instance, the Reactor vein ranges in thickn from 1.5 to 2.0 feet for at least 975 feet of its strike length.

The majority of the thorium-bearing veins strike N40°-50°W or N70°-80°W, although almost every direction except due north is represented. Strike on individual veins commonly varies, but these variations are more apparent on the larger veins. Most of the veins dip steeply to the southwest, but may dip moderately to steeply either to the north or south.

4.4 Geotechnical Data

Based on the very scant information available to date, only the broadest of assumptions can be made with regard to the characteristics of the rock mass in the Lemhi Pass area. It is apparent that the rocks in the area have been disturbed a number of times, and a moderately complex structure exists, probably in conjunction with residual stresses. However, the regional stress field is unlikely to have an overriding adverse effect on mining operations at the shallow depths envisaged.

The effect of the many joints and fractures on the rock mass quality are of great importance to operations and their viability. The data obtained from the three boreholes drilled through the hangingwall, orebody and footwall of the Last Chance

vein indicates the generally low competence of the country
rocks. For example, the Wonder Lode vein lies in a zone of
faulting that affects not only the entire vein along its exposed
length, but also the country rock for as much as 100 feet on
either side (Sharp and Cavender, 1962). It is therefore likely
that considerable support problems could arise in both stopes
and associated excavations. This situation could be
considerably worsened by the presence of an excessive level of
groundwater, particularly in the vicinity of major faults.
There is little evidence at this date, however, to suggest that
such levels of groundwater exist.

4.5 Proposed Exploration Program

To the best of our knowledge, no systematic exploration
program has been carried out in the Lemhi Pass area to
substantiate the existence of economic deposits of thorium. On
the other hand, the information available indicates, that owing
to the methods of exploration used, it is quite possible that
many veins remain undiscovered. In addition to these factors,
geotechnical data must be compiled to better assess the in-situ
ground and water conditions. It is therefore imperative that a
substantial exploration program is initiated prior to
commencement of any mining operation.

The drilling required to prove as orebodies the six veins
selected for mining cost purposes in this report, is a minimum
of 70,000 feet of "NX" diameter diamond drill core. This figure
is based on vertical geological sections at 100-foot centers and
intersecting the veins at 250-foot vertical intervals, with
holes inclined at -60 degrees to the horizontal (see Appendix
A1). A minimum period of two years would be required for the

drilling and data collection program, given reasonable weather
conditions.

5.0   HYPOTHETICAL OREBODY MODEL

A hypothetical model of the likely orebodies can be described as follows:

Deposit Type: Quartz veins and irregular siliceous replacement bodies formed by fissure filling and wall rock replacement along zones of shearing and faulting.

Geometry: Irregular tabular narrow bodies subdivided into veins less than 5 feet wide and veins greater than five-feet wide.   Strike length ranges upwards from 50 feet with some 12 veins greater than 1,000 feet long.   The inclination of both vein classes can be regarded as steep, with an average dip of approximately 60 degrees.   The depth of one vein has been established at about 250 feet below surface.

Grade: Mineralization within the veins is extremely variable.   The mean grade of the Last Chance vein is 0.47 percent ThO_2 which compares with an overall mean for the district of 0.46 percent ThO_2.

Ground Conditions: Overall a poor rock mass quality of both orebody and country rocks.   Stopes and associated tunnels will probably require support.

Water Conditions: Little likelihood of an excessive level of groundwater.

6.0 CONCEPTUAL MINING METHODS

Two mining methods are indicated from the hypothetical orebody model. These are the resue and the cut and fill mining techniques. The resue method is suitable for the extraction of the veins less than 5 feet thick, and the cut and fill technique will satisfy the constraints on mining of the veins greater than 5 feet thick. The stope walls are supported in both methods which, owing to their probable weakness, is a prime consideration for the satisfactory removal of the veins. A distinct advantage of each method is that most of the likely variations in the dip, strike and continuity of the veins can, to a large extent, be included within the boundaries of stopes. The major disadvantages of both methods are economic, in that operating costs are usually high, coupled with low productivities owing to the limited possibilities for mechanization.

6.1 Resuing

Resue mining has generally been used to work very narrow high-grade veins, but its use has diminished over the years. The ore grade and thickness are limiting factors in resuing, since a narrow vein must bear the cost of handling a much greater quantity of waste rock. Another requirement of this method is a well-defined plane of weakness along the contact of the vein and country rock to ensure that the ore and waste can be worked separately with a minimum of dilution. The vein-wall rock contacts at Lemhi Pass are generally sheared.

The sequence of mining resue stopes necessitates the opening of a flat-back overhand stope of minimum width alongside a portion of the narrow vein (Figure 3). The broken waste is

60" DIA. BORED RAISES

ACCESS LEVEL

ORE AND WASTE   IN PLACE

4'x 4' CRIBBED
MANWAY

4'x 4' CRIBBED
CHUTE

BROKEN   WASTE.

HAULAGE LEVEL

|← 150' →| |← 150' →|

0   50   100

SCALE IN FEET

231'

QUARTZ VEIN

2 DRUM SLUSHER HOIST

200'

used to fill the stope for support of the hangingwall and to provide a working base for mining the narrow vein. The broken ore is delivered, probably by scraping, into ore chutes, which are carried through the waste fill. These chutes are extended from the haulage below as mining progresses towards the upper access level.

The layout proposed for the narrow veins at Lemhi Pass is based on 5 feet thick by 300-feet long by 31-feet down dip stopes (Figure 3). Five-foot diameter machine-bored raises are placed in the plane of the vein on 150-foot centers along strike. These raises are utilized for manways, ventilation and ore chutes. Access and haulage levels are situated in the footwall of the veins and are not, as is more usual, developed along the vein. This is done to avoid possible accumulations of radon gas and to seek more competent rock. Proximity of these drifts to veins will be controlled by geotechnical and practical considerations.

Two drum slusher hoists, placed in the center of the stopes, are able to clean either side of a stope to permit a sound operational cycle. A productivity of 4.5 tons per man shift is expected with this method from a two-man crew, based on mucking 34 tons per drilling shift. This rate compares with 25.8 tons mucked per drilling shift in a resue stope at the Tayoltita Mine (Haptonstall, 1978). Assumptions made to determine the expected productivity are enumerated in Appendix A2.

6.2 Cut and Fill

Cut and fill stoping is a suitable method for the extraction of steeply dipping veins greater than 5 feet thick.

Costs estimated for this method have been based on a 10-foot thick vein (see Appendix A3). This technique is currently practiced in one form or another throughout the world. One of the most important aspects of cut and fill stoping, both economically and practically, is the design and implementation of the filling system.

As in resuing, stoping commences from the haulage level and progresses upward. Ore is usually removed in horizontal slices in overhand stopes and delivered to an ore chute which connects to the haulage level below. On completion of each cut, fill is distributed throughout the stope and so provides both support to the hangingwall and a working platform for the next cut.

The design of cut and fill stopes for the Lemhi Pass employs a central 6 foot by 8 foot two-compartment raise (Figure 4). This raise, divided into an ore chute and service compartment, connects the haulage and access levels, which are 200 feet apart, vertically. The assumed stope length of 300 feet along strike could alter, depending upon varations in the strike length and width of the vein.

An air-operated loader is used in the stope to haul the broken ore to the central ore chute. For stopes less than 10 feet wide, a slusher would be utilized. It is estimated that a three-man crew can produce at a rate of 116 tons per shift.

6.2.1 Backfill System

It is intended to obtain fill material from adjacent Quaternary deposits as opposed to using deslimed mill tailings, which have a potential for releasing radon gas into the mine airstream. Thus, the fill material is a high-cost item,

IDEALIZED CUT AND FILL STOPE

Figure 4

ACCESS LEVEL

ORE IN PLACE

WASTE PILLAR OR FILL FENCE

300'

8'

SAND FILLED STOPE

CHUTE → ← MANWAY

WASTE PILLAR OR FILL FENCE

HAULAGE LEVEL

0   50   100

SCALE IN FEET

PILLAR

231'

6' X 8' TWO COMPARTMENT
RAISE - STULLED & LAGGED

10'

200'

6' X 8' TWO COMPARTMENT
CRIBBED RAISE

particularly if the handling problems expected in cold weather are considered. It has been assumed that sand is delivered to the mine site by a subcontractor at a cost of $5.00 per cubic yard, including stripping the sand deposit, mining, reclamation, hauling, processing, royalties, and protection of the sand during cold weather. Processing of the sand at the mine site involves screening off the plus 1/8-inch fraction and would also include removal of any clay materials.

The fill system envisaged consists of a water tank, agitator tank, cement hopper and, for the lower blocks, a 5 foot diameter bored storage raise (Figure 5). For the upper blocks, sand is conveyed to an agitator tank where it is mixed with water to form a 60 percent solids slurry. Sufficient water for this system could be pumped from the local streams. From the agitator, hydraulic fill passes to the stopes through a 3 inch diameter borehole and pipe system. For the lower blocks, the slurry would be mixed underground adjacent to the storage raise. The final 6 inches of each fill lift would consist of a sand and cement mix to provide a satisfactory working surface for the efficient working of the rubber-tired loader.

6.3 Overall Mine Design

The basis for the cost estimates for mining the thorium-bearing veins in the Lemhi Pass area are outlined in this section and are detailed in Appendix A. The constraints of vein geometry, topography and environmental considerations limit both the mine entrance location and potential size of possible mining operations in this district. The site for surface mine buildings exists adjacent to the south fork of the Agency Creek

Sand Deposit

End Dump Trucks

Mine Site

Screen ⟶ + ⅛" Fraction

– ⅛" Fraction

Hopper

Water Tank
50,000 gallon

10" dia.
Borehole

Agitator
Tank
85,000 gallon

Cement
Hopper

Storage
Raise

60"
dia.

3"dia. Borehole
and Pipeline

3"dia.
Pipeline

Stope

Stope

road (Figure 2). The Tendoy Copper Queen Mine and the Wonder
Mine were both served by this road.

The portal of the major adit would therefore be sited in
this vicinity, which has the added advantage of being only 6,600
feet above mean sea level. This is one of the lowest elevations
from which to gain access to the greater part of the lower
blocks of the veins selected for mining. The direction of this
adit would be approximately due south at a gradient of +1/2
percent (Figure 2), thus crossing the strike of the known veins
at approximately right angles and giving the best chance of
exposing additional mineralization. This type of major access
is preferred to establish an early cash flow. It offers an
apparent cost advantage relative to shaft access, speed of
development compared to shaft sinking, cost savings afforded by
the use of gravity for ore transportation and drainage, and
relative ease and flexibility associated with ventilation. A
further four adits are planned to provide access to the upper
sections of the lower blocks at 200-foot intervals up to the
7,400-foot level. The upper blocks will also be accessed by a
further eight adits.

A production rate of 1,000 tons per day is thought to be a
target that can be reasonably attained. Ordinarily, various
production capacities would be analyzed and the capacity
demonstrating the highest economic return chosen. However, this
degree of refinement has not been possible within the scope of
this report, and with the restricted geological information
available.

Calculations to establish a hypothetical ore reserve were
made on six veins selected for their potential mineability. A
reserve of approximately 3.5 million tons was estimated from the

data available, assuming that the veins extended in depth to the level of their respective access adits, and were inclined at the angle of dip measured at their outcrop. Based on these major assumptions, the potential life of the mine is about 14 years at a production rate of 250,000 tons per year (250 days per year; two shifts per day), an estimated mining recovery of 90 percent, and dilution rates of 20 percent in resuing and 10 percent in cut and fill. The relative rates of production for the resue and cut and fill methods are 20,000 and 230,000 tons per year, respectively during the first four years of production (Figure 6). These change slightly to 10,000 and 240,000 tons per year, respectively, during the fourth year of production and remain at this level until the projected end of operations.

Following the completion of the exploration program and feasibility study, predevelopment of the mine is initiated some eight months prior to the startup of production (Figure 7). This development consists of 4,825 feet of trackless drifts and 1,650 feet of bored and stulled raises for the upper blocks. An additional 1,800 feet of tracked drift is required for the lower blocks. Thus, 12 stopes would be prepared for production to ensure the maintenance of a continuous flow of ore to the mill.

Trackless drifts in conjunction with 10-ton tip-dump trucks were adopted for hauling ore from the upper blocks because of the apparent cost advantage of this system relative to tracked drifts. The advantage of rubber-tired equipment derives from the short hauls involved (average of 1,500 feet), its flexibility, versatility, simplicity, and the lower tonnages available in the upper blocks. Conversely, these distances are too great for a reasonably sized load-haul-dump unit. The tracked haulage favored for the greater haul distances and tonnages of the lower blocks consists of a 10-ton locomotive and

Item	Quantity	Unit	Cost Item	0 REVENUE YEARS
				-1, 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 13, 14, 15
PRODUCTION SCHEDULE				
Resue Stopes – Upper blocks 20,000 TPY	65,000	T	215	bar years 1–4
Resue Stopes – Lower blocks 10,000 TPY	99,000	T	215	bar years 4–13
Cut and Fill Stopes – Upper blocks 230,000 TPY	745,000	T	210	bar years 1–4
Cut and Fill Stopes – Lower blocks 240,000 TPY	2,635,000	T	210	bar years 4–14
TRACKLESS HAULAGE	810,000	T	260	bar years 1–4
RAIL HAULAGE	2,734,000	T	270	bar years 4–14
DEVELOPMENT SCHEDULE Upper Blocks				
11' x 10' Trackless Drift 7000 LF/YR	11,825	LF	220	bar years -1–1
6' x 8' Stulled Raise 2750 LF/YR	3,200	LF	241	bar year -1–1
5' dia. Bored Raise 6250 LF/YR	4,950	LF	240	short bars year -1 and year 1
N.B. 23 Chutes and 8 Portals to be installed				
Lower Blocks				
10' x 10' Adit – 6600 Track 2000 LF/YR	9,400	LF	230	bar years -1–3
10' x 10' Adit – 6800 Track 2000 LF/YR	8,000	LF	230	bar years 1–3
10' x 10' Adit – 7000 Track 2000 LF/YR	7,000	LF	230	bars years 2–3 and years 5–6
10' x 10' Adit – 7200 Track 2000 LF/YR	6,400	LF	230	bars years 2–3 and years 5–6
11' x 10' Adit – 7400 Trackless 3500 LF/YR	3,000	LF	220	bar year 6
10' x 10' Orebody Drifts Track 4000 LF/YR	17,800	LF	230	bar years 3–6
6' x 8' Stulled Raise 2750 LF/YR	11,600	LF	241	bar years 4–5
5' dia. Bored Raise 6250 LF/YR	6,700	LF	240	bar year 5
N.B. 77 Chutes and 5 Portals to be installed				

Figure 6.

Quantity	Unit	Cost Item	REVENUE YEARS
			0, -1, 1, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 13, 14, 15
65,000	T	215	bar: year 1–4
99,000	T	215	bar: year 4–13
745,000	T	210	bar: year 1–4
635,000	T	210	bar: year 4–14
810,000	T	260	bar: year 1–4
734,000	T	270	bar: year 4–14
11,825	LF	220	bar: year -1–1
3,200	LF	241	bar: year 0–1
4,950	LF	240	bar: year 0, year 1
9,400	LF	230	bar: year -1–4
8,000	LF	230	bar: year 0–4
7,000	LF	230	bar: year 2–3, year 5–6
6,400	LF	230	bar: year 2–3, year 5–6
3,000	LF	220	bar: year 5–6
17,800	LF	230	bar: year 3–7
11,600	LF	241	bar: year 3–5
6,700	LF	240	bar: year 3–4

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PRELIMINARY FEASIBILITY STUDY ← Go/No Go Decision

EXPLORATION PROGRAM

FEASIBILITY STUDY
    Sampling & Metallurgical Tests

    Engineering - Definitive ← Go/No Go Decision

ENVIRONMENTAL

ENGINEERING - Design

MINE DEVELOPMENT
    Prerevenue
PREREVENUE

-5   -4   -3   -2   -1   0   +1
Years

Project No. _____ Reviewed _____ Date _____

D.31

10 cars of 7 ton capacity. Width and height of the tracked and trackless drifts for the necessary haulage equipment and ventilation are 10 ft by 10 ft and 11 ft by 10 ft, respectively. Two boom drilling jumbos with 10-foot feeds will be used for the development of the drifts. A 2 cubic-yard load-haul-dump unit with an ejector bucket will clean the trackless drift and an air-powered overshot mucker with conveyor belt will clean the development muck from the tracked drift.

Development costs for the various drifts have been estimated exclusive of support costs (Appendix A6). If ground conditions are such that support is required, as is thought most likely, the cost of support will increase the overall development cost about $50 per foot. Some form of shotcrete lining applied by a dry mix shotcrete machine would provide a satisfactory solution to most of the support problems envisaged.

The ventilation of a mine subjected to emission of radon gas requires special attention in its design and application (Rock, 1969, and Schroeder and Evans, 1969). The most important factors accouring to Rock (1969) are to:

1. Minimize recirculation.
2. Drive intake air headings (adits, haulages) in barren ground.
3. Maintain fans running during off shifts.
4. Keep mined-out areas on the return side of the ventilation circuit.
5. Drive drifts as large as possible to minimize air resistance.
6. Keep air velocities through the workings at a minimum flow of 100-feet per minute.

A push-pull system of ventilating the numerous working areas,
using secondary fans maintained at a high standard, is
considered most suited to dealing with the emission of radon gas
from the exposed veins and broken ore. Provision has been made
in the project cost to heat the ventilating air during the
winter months.

Given the basic configuration of the mine and the produc-
tion rate of 1,000 tons per day, total manpower requirements
have been estimated at 142, with 112 men required for the
operation of the mine, 17 men employed on maintenance duties,
and 13 staff to carry out administrative and supervisory roles
(Table 2).

TABLE 2

MANPOWER REQUIREMENTS AND COSTS AT 250,000 TONS PER YEAR (250 DAYS)

ITEM	$/HR.	$/Shift	Number Per Shift	Shifts Per Day	Total
OPERATING LABOR					
Mine					
Miners	11.25	90.00	20	2	40
Fill Crew	7.13	57.00	3	2	6
Miners-Develop	11.25	90.00	14	2	28
Diamond Drillers	11.25	90.00	2	2	4
10-Ton Truck Operators	11.25	90.00	3	2	6
Loco Operators	11.25	90.00	2	2	4
Timber/Pipe/Vent. Crews	9.75	78.00	6	2	12
Track/Tool/Supply & Labor Crews	7.13	57.00	6	2	12
				Sub-Total	112
Maintenance					
Electricians	11.25	90.00	1	2	2
Mechanics	11.25	90.00	2	2	4
Lampman	7.50	60.00	1	1	1
Drill Doctors	9.75	78.00	1	2	2
Supply/Labor Crews	7.13	57.00	4	2	8
					17

ADMINISTRATIVE & SUPERVISORY			Total	$/Year
Mine Manager	1 @	$40,000	1	$ 40,000
Superintendent	1 @	31,000	1	31,000
Foreman	4 @	25,000	4	100,000
Engineer	1 @	31,000	1	31,000
Geologist	1 @	22,000	1	22,000
Admin. Manager	1 @	25,000	1	25,000
Timekeeper/Warehouse	1 @	20,000	1	20,000
Surveyor	1 @	16,000	1	16,000
Secretary/Clerks	2 @	10,000	2	20,000
Subtotal			13	$305,000
Fringes @ 30%				91,500
Total with Fringes				$396,500/YR

MANPOWER GRAND TOTAL   142

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7.0 PROJECT COSTS

The capital and operating costs outlined in this section
are founded on the proposed development of a mining complex and
infrastructure to mine six thorium-bearing veins with an
estimated hypothetical ore reserve of approximately 3.5 million
tons. These costs are related to the mining costs, inclusive of
water and waste management but exclusive of the cost of milling,
environmental considerations, disposal of mill tailings, land
acquisition and royalties.

7.1 Exploration and Prerevenue Expenditure

The main stages of the exploration and prerevenue program
include the preliminary feasibility study, drilling program,
feasibility study, engineering design, mine development and mill
construction (Figure 7). During the later stages of the
exploration program, assuming that initial ore reserve
calculations are encouraging, a sampling and metallurgical
testing program should be initiated. The data developed from
these tests will provide information relative to milling
characteristics, recoveries, costs and design parameters.

With regard to environmental considerations, it may be
expedient to provide data for purposes of aiding in the prepara-
tion of impact statements. Such data may also be useful to con-
cerned agencies as an indication that adequate consideration to
the environment has been given. The cost and timing of this
expenditure are dependent on the particular local environment,
location, proposed extent of operations and political climate.
Consequently, the magnitude of the cost and timing of this
effort are difficult to estimate and have not been attempted.
Careful analysis of the preliminary environmental data should be

conducted prior to any significant expenditure relative to continuation of the exploration program and feasibility study.

Expenditure for the exploration program amounts to $1,915,000, which is just over half the total prerevenue expenditure of $3,442,000 (Table 3). An allowance of $100,000 is made for the feasibility study performed by a firm of consultants and an additional lump sum of $150,000 is appraised for the engineering design work. The other major cost item prior to production is the prerevenue development cost at $1,267,000 (Table 4). This sum includes $570,675 for 6,625 feet of tracked and trackless drifts, $164,130 for 1,650 feet of raise development and chute installations and $450,550 for indirect charges such as staff salaries, utilities, transport and personnel acquisition and training (Table 4). Minor provisions of $24,000 and $42,420 have been made for access to and construction of the portals for the upper and lower blocks. The derivations of the units used to calculate the total cost of the different items are detailed in the Appendix.

During this prerevenue period, only nine of the 13 staff necessary for full production are required. Thus, only nine staff members have been accounted for in the eight-month period prior to startup of production. A significant contribution to the cost set against utilities, etc., is made by the operating cost of the transport facilities which are necessary to convey personnel and materials from Salmon and Dillon.

7.2 Operating Costs

The materials and supply costs for each piece of mining and ancillary equipment were computed based upon the projected productivity of the types of equipment and manpower specified by the mining schedule.

TABLE 3

ESTIMATED EXPLORATION AND PREREVENUE   EXPENDITURES

| ITEM | PROJECT YEAR AND COSTS | | | | | TOTAL |
	-5	-4	-3	-2	-1	COST $
Preliminary Feasibility Study	10,000					10,000
Exploration Program						
Consulting Geologists	30,000	60,000	30,000			120,000
Drilling	395,000	788,000	395,000			1,578,000
Other	54,000	109,000	54,000			217,000
Feasibility Study Engineering						
economic, marketing, transport, etc.			100,000			100,000
Engineering-Design				100,000	50,000	150,000
Prerevenue Development					1,267,000	1,267,000
	489,000	957,000	479,000	100,000	1,317,000	3,442,000

TABLE 4
PREREVENUE   DEVELOPMENT

ITEM		UNITS	COSTS ($)			
			LABOR	SUPPLIES	TOTAL	UNIT
Trackless Drifts	11' x 10'	4,825'	149,575	212,300	361,875	75/ft.
Tracked Drifts	10' x 10'	1,800'	102,600	106,200	208,800	116/ft.
Stope Development & Preparation						
Bored Raise	60" dia.	1,250'	30,000	50,000	80,000	64/ft.
Stulled Raise	6' x 8'	400'	14,800	13,600	28,400	71/ft.
Chutes		12			55,730	4,644 ea.
Indirects (8 months)						
Staff Salaries					150,000	
Utilities, Transport, Power					180,550	
Personnel Acquisition and Training					20,000	
Access Roads (to portals and levels)		10,000'			24,000	2.4/ft.
Site Preparation Portals		9			42,420	4,713 ea.
Subtotal					1,151,775	
Contingency					115,225	
TOTAL PREREVENUE DEVELOPMENT					$1,267,000	

Supplies are specified as items consumed by or installed on the mechanized equipment, such as fuel, electric power, tires, parts, lubricants, drill steel and bits. These costs were assigned either as an operating cost, hourly, per shift, or yearly per operating unit; or, as in the case of drill steel and bits, on a unit footage or tonnage basis. The buildup of machine operating costs generally follow manufacturers' recommendations, except in cases where operating experience or background indicate a more conservative estimate is appropriate. In most cases a more conservative, and hence generally higher, operating cost was derived than that which would be obtained by strictly following manufacturer assessments.

The costs of labor and staff are detailed in Table 2 and reflect the likely rates of pay in the states of Montana and Idaho. The hourly cost given for the operating labor is inclusive of fringe benefits and bonus payments.

Operating costs for the 14-year life of the mine are summarized in Tables 5 and 6. During the first seven years of production the operating cost per ton of ore is $22.17 compared to a cost of $13.92 per ton for the final seven years. Over the life of the mine, the average operating cost per ton is $18.00. The higher initial operating cost reflects the cost of development and exploration, which are completed during the first seven years of production. Indirect costs also contribute to reduced costs during the final phase of operations since fewer maintenance and supervisory personnel are required. A detailed breakdown of these costs is included in the Appendix. The total operating cost for the producing life of the mine is $63,781,234.

TABLE 5

ESTIMATED OPERATING COSTS TO PRODUCE 1,750,000 TONS,
YEAR 1 to 7 INCLUSIVE

ITEM	UNITS	COSTS ($)			
		LABOR	SUPPLIES	TOTAL	UNIT
Stoping					
Resue	102,000 T	2,305,200	979,200	3,284,400	32.20
Cut and Fill	1,648,000 T	4,663,840	2,735,680	7,399,520	4.49
Stope Fill	732,000 CY	945,000	4,867,800	5,621,760	7.68
Stope Chutes	100 EA	200,000	264,400	464,400	4,644.00
			Sub-Total	16,770,080	$ 9.58/T
Development					
11'x10' Trackless	10,000 LF	310,000	440,000	750,000	75.00
10'x10' Tracked	46,800 LF	2,667,600	2,761,200	5,428,800	116.00
6'x8' Raise	14,400 LF	518,400	489,600	1,008,000	70.00
5' Dia. Raise	10,400 LF	249,600	416,000	665,600	64.00
Portals	3 EA	11,415	2,724	14,139	4,713.00
			Sub-Total	7,866,539	$ 4.50/T
Haulage					
Rubber Tired	810,000 T	510,300	202,500	712,800	0.88
Track	940,000 T	479,400	131,600	611,000	0.65
			Sub-Total	1,323,800	$ 0.76/T
Exploration					
Diamond Drilling	62,300 LF	668,479	723,926	1,392,405	22.35
			Sub-Total	1,392,405	$ 0.80/T
Indirect Costs					
Underground	PER YEAR	438,750	241,650	4,762,800	680,400/YR
Surface	PER YEAR	179,000	330,300	3,565,100	509,300/YR
Admin. & Supervisory	PER YEAR	396,500		2,775,500	396,500/YR
Insurance	PER YEAR		50,000	350,000	50,000/YR
			Sub-Total	11,453,400	$ 6.54/T

TOTAL OPERATING COSTS FOR YEAR 1 TO 7 INCLUSIVE = $38,806,224

Cost per ton $22.17

D.40

TABLE 6

ESTIMATED OPERATING COSTS TO PRODUCE 1,794,000 TONS,
YEAR 8 TO 14.18 INCLUSIVE

| ITEM | UNITS | COSTS ($) | | | |
		LABOR	SUPPLIES	TOTAL	UNIT
Stoping					
Resue	62,000	1,401,200	595,200	1,996,400	32.20
Cut and Fill	1,732,000	4,901,560	2,875,120	7,776,680	4.49
Stope Fill	750,400	772,912	4,990,160	5,763,072	7.68
			Sub-Total	15,536,152	$8.66/T
Haulage					
Track	1,794,000	914,940	251,160	1,166,100	0.65
			Sub-Total	1,166,100	$0.65/T
Indirect Costs					
Underground	PER YEAR	303,745	222,920	3,781,454	526,665
Surface	PER YEAR	100,500	220,030	2,301,405	320,530
Administrative & Supervisory	PER YEAR	255,000		1,830,900	255,000
Insurance	PER YEAR		50,000	359,000	50,000
			Sub-Total	8,272,759	$4.61/T

TOTAL OPERATING COSTS FOR YEAR 8 TO 14.18 INCLUSIVE = $24,975,010

Cost per ton $13.92

7.3 Capital Costs

Initial capital expenditure on equipment and buildings amounts to $3,614,000 (Table 7). The total cost of replacing equipment during the revenue years of the mine is $552,500.

Operating, capital, replacement, depreciation, prerevenue expenditure and amortization costs are summarized in a schedule of estimated annual expenditure shown in Figure 8.

7.4 Upper and Lower Bound Costs

The upper and lower bounds and estimated costs for the project are summarized in Table 8.

TABLE 8

SUMMARY OF UPPER AND LOWER BOUND AND ESTIMATED PROJECT COSTS

Cost Center	Lower Bound	Estimated Costs $	Upper Bound
Capital	3,253,000	3,617,000	5,100,000
Replacement	507,000	552,500	780,000
Prerevenue	3,240,000	3,442,000	8,000,000
Total Operating	57,400,000	63,781,234	91,120,000
	64,400,000	71,389,734	105,000,000

The upper bound estimated cost of $105,000,000 is based on the assumptions that ground conditions could be extremely difficult and that the geometry of the veins varies considerably. The cost of the exploration program could therefore conceivably triple. Poor ground conditions would also seriously affect both

Item	Quantity	Unit	Cost Item	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
OPERATING COSTS See Table 5 & 6	63,781,234	$							5543750	5543750	5543750	5543750	5543750	5543750	5543724	3478000	3478000	3478000	3478000	3478000	347800
CAPITAL COSTS See Table 7	3,614,000	$						3614000													
REPLACEMENT COSTS See Table 7	1,985,500	$								14000	18000	466000	402000	32000	107500	254000	18000	402000		272000	
DEPRECIATION See Table 7	5,599,500	$							552500	552500	552500	552500	434500	434500	434500	366830	276330	276330	295730	295730	28752
PREREVENUE EXPENDITURE See Table 3	3,442,000	$		489000	957000	479000	100000	1317000													
AMORTIZATION OF PREREVENUE EXPENDITURE	3,442,000	$							245860	245860	245860	245860	245860	245860	245860	245860	245860	245860	245860	245860	24586
PRODUCTION TONS See Table A4	3,544,000	T							250000	250000	250000	250000	250000	250000	250000	250000	250000	250000	250000	250000	25000
GRADE % ThO₂ See Table A4	0.26/0.28	%							←	0.26	→	←	→				0.28				
ThO₂ Tons	9763.5	T							650	650	650	687.5	700	700	700	700	700	700	700	700	70

Quantity	Unit	Cost Item	-5	-4	-3	-2	-1	0/1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
3,781,234	$							5543750	5543750	5543750	5543750	5543750	5543750	5543724	3478000	3478000	3478000	3478000	3478000	3478000	3478000	629010
3,614,000	$						3614000															
,985,500	$								14000	18000	466000	402000	32000	107500	254000	18000	402000		272000			
5,599,500	$							552500	552500	552500	552500	434500	434500	434500	366830	276330	276330	295730	295730	287525	287525	
3,442,000	$		489000	957000	479000	100000	1317000															
3,442,000	$							245860	245860	245860	245860	245860	245860	245860	245860	245860	245860	245860	245860	245860	245860	245820
5,544,000	T							250000	250000	250000	250000	250000	250000	250000	250000	250000	250000	250000	250000	250000	250000	45000
0.26/0.28	%							← 0.26 →			←					0.28						→
9763.5	T							650	650	650	687.5	700	700	700	700	700	700	700	700	700	700	126

TABLE 7
ESTIMATED CAPITAL EXPENDITURE AND ANNUAL DEPRECIATION

ITEM	UNITS	COST ($) UNIT	COST ($) TOTAL	LIFE YEARS	ANNUAL DEPRECIATION $
LHD - 2 Cubic Yard	1	78,000	78,000	7	11,143
Truck 10T	3	124,000	372,000	7	53,142
Jumbo 2 Boom	1	60,000	60,000	8	7,500
Jumbo 2 Boom (Rail)	4	60,000	240,000	8	30,000
Jackleg	6	3,000	18,000	5	3,600
Stoper	12	2,500	30,000	5	6,000
Raise Borer	1	472,000	472,000	4	118,000
Loader 1-3/4T	10	34,000	340,000	5	68,000
Slusher 15 HP	5	6,000	30,000	14	2,143
Mucker	4	53,000	212,000	4	53,000
Trolley Loco 10T	1	82,500	82,500	14	5,893
Trolley Loco 8T	1	66,000	66,000	14	4,714
Granby Car	20	7,400	148,000	14	10,571
Shotcrete M/C	2	7,000	14,000	5	2,800
Diamond Drill	1	8,500	8,500	7	1,214
Tugger Hoist	15	1,500	22,500	7	3,214
Ventilation Ducting	16,000	0.50	8,000	2	4,000
Boss Buggy	1	27,000	27,000	7	3,857
Fan 20 HP/10,000 CFM	18	2,500	45,000	7	6,429
Fan 10 HP/5,000 CFM	7	2,000	14,000	7	2,000
Compressor 10,000 CFM	1	300,000	300,000	14	21,429
Bus 45 Seat	3	80,000	240,000	4	60,000
5T Truck	1	14,000	14,000	7	2,000
Pickup	3	6,000	18,000	3	6,000
Car	1	6,000	6,000	2	3,000
Offices			40,000	14	2,857
Workshops and Equipment			145,000	14	10,357
Compressor House			40,000	14	2,857
Utilities			125,000	14	8,928
Powder Magazine			30,000	14	2,143
Mine Dry			40,000	14	2,857
			3,285,500		519,648
Contingency			328,500	14	32,850
	TOTAL		3,614,000		552,498

development and stoping costs. Variable vein geometry would
result in increased exploration and development requirements and
hence an overall increase in costs.

7.5 Sensitivity Analysis

A sensitivity analysis of three parameters has been carried
out in order that the estimates made in this study may be
applied to similar vein deposits. These parameters are grade,
vein width, and ore reserve; the basis on which these analyses
were made are:

I)   The effect of a change in vein width is related to the
     cost per pound of ThO_2 per stope and not to the
     overall cost of the mine.

II) The effects of changes in mine grade and ore reserve
     are related to the cost per pound of ThO_2 inclusive
     of the overall cost of the mine.

III) The costs of development for standard resue and a cut
     and fill stopes per 300' strike length and 231' dip
     length are 56,712 dollars and 43,314 dollars,
     respectively. These costs include 4,644 dollars for
     the installation of chutes.

7.5.1 Grade Variation

The total cost of ThO_2 produced from the proposed mine
is 3.66 dollars per pound at a grade of 0.28 percent ThO_2.
Grade is directly related to the cost per pound of ThO_2
(Table 9).

TABLE 9

EFFECT OF A VARIATION IN MINE GRADE
ON THE COST OF ThO_2

Grade Percent	Lb. ThO_2 Per Ton	COSTS/LB. ($)		
		Operating	Capital	Total Cost
0.1	2	9.00	1.08	10.08
0.2	6	3.00	0.36	3.36
0.5	10	1.80	0.22	1.44
0.7	14	1.29	0.15	1.12
0.9	18	1.00	0.12	1.01
1.0	20	0.90	0.11	0.50
2.0	40	0.45	0.05	

Thus, if the average grade of ore varied from 1 percent to 0.1 percent, the cost per pount of ThO_2 produced would vary between 1.01 dollars and 10.08 dollars, respectively.

7.5.2 Vein Width Variation

A change in the width of veins has a significant effect on the cost per pound of ThO_2 produced by the resue stoping method (Table 10). The grade assumed for the purpose of analyzing the sensitivity of the cost of ThO_2 to vein width is 0.28 percent or 5.6 lb. per ton of ore.

TABLE 10

EFFECT OF A VARIATION IN VEIN WIDTH
ON THE COST OF ThO_2

MINING METHOD	VEIN WIDTH (ft)	COSTS/LB. ($)			
		FILL	DEVELOPMENT	STOPE	TOTAL
	0.75	–	2.34	11.50	13.84
	1.50	–	1.17	5.75	6.92
	2.25	0.37	0.78	3.83	4.98
	3.00	0.48	0.58	2.88	3.94
	4.50	0.58	0.39	1.92	2.89
All Resue	5.00	0.61	0.35	1.67	2.63
	10.00	0.61	0.13	0.80	1.54
	15.00	0.61	0.09	0.80	1.50
	20.00	0.61	0.07	0.80	1.48
All Cut and Fill	30.00	0.61	0.04	0.80	1.45

From Table 10 it can be seen that it is necessary to introduce fill into the resue stopes with vein widths of 2.25 ft, and both development and stoping costs greatly affect the cost of ThO_2 in resue stopes with vein widths up to 5 ft. Thereafter, in cut and fill stopes, only the cost of development changes by a small amount. Only stope development is included for this analysis. Thus, for a variation in vein width from 0.75 ft. to 30 ft, the cost of ThO_2 changes from 13.84 dollars to 1.45 dollars per pound, respectively.

7.5.3 Ore Reserve Variation

The effect of a variation in ore reserve on the cost of a pound of ThO_2 is controlled by a number of factors. For example, the distance between the vein deposits of ThO_2 can require extensive development to expose even decreasing amounts of ore of diminishing grade. The datum for this sensitivity analysis is the ore reserve of 14,161 tons as indicated in

Table A4.1, of which only 9761 tons can be mined. The maximum additional ore reserve available for this analysis is some 23,751 tons of ThO_2, as detailed in Table A4.2, of which only 13,993 tons can be extracted. The effect on the cost of ThO_2 of mining this extra ore is enumerated in three stages (Table 11). Other veins in the district that contain thorium have been studied for this analysis, such as the Lucky Horseshoe and the Buffalo veins, but have no- been used, as a shaft is required for their development. It has not been possible to account for the cost of a shaft within the time constraints of this report. The effect on the cost of ThO_2 by sinking shaft is difficult to assess without a detailed study, for while capital expenditure is necessary for its development, operating costs would be increased by hoisting operations. The ore reserve could be improved not only from the Lucky Horseshoe veins but also from such veins as the Wonder Lode and Last Chance veins.

TABLE 11

EFFECT OF A VARIATION IN MINING
RESERVE ON THE COST OF ThO_2

Mining Reserve Tons of ThO_2	Mean Grade %	COST			
		Capital	Operating	Total	$/LB
9,761	0.28	7,611,500	63,781,234	71,392,734	3.66
11,762	0.40	3,000,000	84,084,000	87,084,000	3.70
413	0.51	20,000	4,200,000	4,220,000	5.19
1,818	0.46	1,000,000	18,350,000	19,350,000	5.32

It is evident from Table 11 that the estimated cost per pound of thorium increases from 3.66 dollars to 5.32 dollars as mining reserves increase from 9761 tons to 23,754 tons, respectively. The major factors which contribute to this

increase are the distance of the veins from the mine site in
Agency Creek and that the final 2200 tons of ThO_2 are extracted
by resuing methods. Further, the high capital cost to remove
the final 2200 tons is necessitated by the purchase of addi-
tional haulage trucks and road construction. A haulage cost
of 2.00 dollars per ton was estimated to move the ore from
the Bear and Trapper Creeks to Agency Creek. Other cost
factors for this analysis were take from Tables 5, 6, and 7.

Respectively submitted,

GOLDER ASSOCIATES, INC.

B. A. Kennedy, P.E.

W. J. Brown, P.E.

for . G. A. Ferguson

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APPENDIX A
UNDERGROUND MINING OF THORIUM DEPOSITS, LEMHI PASS

The basic data utilized to determine the costs given in this report are detailed in this appendix.

A.1 DIAMOND DRILLING

Data

1. Geological vertical sections required on 100-foot centers.
2. Three holes drilled per section, at -60° to intersect vein at vertical depths of 250 feet, 500 feet and 750 feet below surface.
3. Length of drilling per section = $\dfrac{250' + 500' + 750'}{\sin 60°}$ = 1730'
4. Forty sections drilled for feasibility study, or about 70,000 feet. A further 36 sections drilled from startup of production to year 7, approximately 62,300 feet.
5. Advance rate of 20 feet per shift, i.e., 25 shifts for 500 foot hole plus three shifts moving machine.

Costs - Item 280

1. Average drilling cost per foot in 1976 - $14.50 exclusive of ancillary charges.
2. Inflation discounted at six percent per year and ancillary charges at 38 percent.
3. 1978 drilling cost = $14.50 x 1.38 x 1.12 = $22.35, made up of $10.73 labor and $11.62 supplies.
4. Estimated life of drill bit, 500 feet, at a cost of $10 per foot.

A.2 RESUE STOPING

Data

1. Slushing limit of 150 feet to give stope length of 300 feet.
2. Single central raise of stopes, raise bored to provide adequate ventilation.

3. Stope thickness 5 feet: waste 3.5 feet, ore 1.5 feet.

4. Drilling efficiency 35 feet/man-hour.

5. Drilling required for 24T of waste and 10T ore per shift, is 140 feet and 90 feet respectively. Length of round along strike 15 feet drilled with 6-ft steels.

6. Total drilling time for driller $\underline{\frac{140' + 90'}{35'/MH}} = \underline{6.6 \text{ man-hours.}}$

7. Average cycle time for 10 H.P. slusher over 75-ft. pull with 30-inch slusher is 1.10 minutes per 627 pound load in iron ore.

   Equivalent in quartz 3.8 min/T,

   i.e., 2.2 Hr. for slushing per shift.

8. Shift work of second crew member:

   2.2 Hr. – Slushing

   1.0 Hr. – Trimming

   3.2 Hr. – Loading holes

   $\underline{0.2 \text{ Hr.}}$ – Utilities

   $\underline{6.6 \text{ Hr.}}$

9. Each 12-foot lift, four crew shifts are estimated to move cribbing and slusher.

10. Ore tons per stope lift = $\dfrac{300' \times 1.5' \times 12'}{12 \text{ ft}^3/T} = \underline{450T}$

11. Productivity of two-man crew = $\dfrac{450T}{10T}$ = 45 crew shifts + 4 crew shifts moving to other stopes, therefore $\dfrac{450T}{49 \text{ crew shifts}}$

    =9T/shift or $\underline{4.5T/\text{man-shift.}}$

12. Slusher repair costs estimated at $5.00 per hour for parts and supplies. Steel rope costs figured at $1.00 per foot, or $300 per stope. Life estimated at 40 shifts, thus cost per shift is $7.50.

13. Explosives use estimated at 2.5 lb. per ton of ore for 75 percent ANFO – 25 percent Dynamite.

Costs - Item 210

1. The contribution of labor and supplies to the cost of
   $32.20/ton of ore by resue stoping is given in Table A1.
2. Average explosives cost, including caps, is $0.63 per ton
   of ore broken.

A.3 CUT AND FILL STOPING

Data

1. Standard stope thickness - 10 feet.
2. One 10-ft. breast round drilled per shift by two men, third
   crew member mucks and charges drill holes. Area of rock
   per round is 78 square feet. Thus, number of holes per
   round = $\frac{78 \text{ sq.ft.}}{3 \text{ sq.ft./hole}}$ = 26. 33 rounds needed per lift.
3. Tonnage broken per round - 58 tons.
4. Central ore pass required per 300-foot long stope.
5. Productivity of three-man crew - 116 tons per shift.
6. Stope volume = $\frac{300' \times 10' \times 200'}{\sin 60°}$ = $\underline{693,000 \text{ ft}^3}$

   therefore, tons per stope = $\frac{693,000 \text{ ft}'3}{12 \text{ ft}^3/T}$ = $\underline{57,700 \text{ tons}}$
7. Stopes required to produce 900 tons per day =

   $\frac{900}{116}$ = $\underline{8 \text{ stopes}}$
8. Powder factor - 2 lb. per ton of ore or <u>116 lb. per round.</u>
9. 75 board feet of cribbing placed per foot of raise.
10. Estimated productivity of air loader - 40 tons per hour.
    Muckout time $\frac{58T}{40 \text{ TPH}}$ = 1.5 Hr for 150' tramming distance.

Costs - Item 210

1. The breakdown of labor and supplies towards the $4.49 cost
   per ton ore is detailed in Table A2.
2. Miscellaneous stope timber is not included in direct stope
   costs.

TABLE A1
ESTIMATED COST OF RESUE STOPING

Cost Item	Units	Labor	Supplies	Total	Unit Cost
Miners 4.5 T/MS	2.22 MS	200		200	90
Bits & Steel	2.30 LB		23	23	0^{10}
Explosives	85 LB		21	21	0^{25}
Raise Cribbing	0.28 LF		21	21	37^{34}
Stopers (4)	6.60 Hr	20	23	43	6^{50}
15 HP Elec. Slusher	2.20 Hr	6	8	14	6^{00}
Cost/Round		226	96	322	
Cost/Ton		22^{60}	9^{50}	32^{20}	

NOTE: Basis of estimate is one round in ore plus removal of $3\frac{1}{2}$ ft. thickness of waste. This would yield 10 tons of ore. Figure 1 round is 15 L.F. along strike, drill 6 ft, pull $5\frac{1}{2}$ ft. This would be one shift work for 2 men, 10T/crew-shift.

TABLE A2

ESTIMATED COST OF CUT AND FILL STOPING

Cost Item	Units	Labor	Supplies	Total	Unit Cost
Miners 38.7 T/MS	1.50 MS	135		135	90
Bits & Steel	260 LF		26	26	0^{10}
Explosives	116 LB		29	29	0^{25}
Raise Cribbing	0.28 LF		10	10	37^{34}
Jacklegs (4)	6.50 HR	20	23	43	6^{50}
$1^3/_4$T R.T. Loader	1.50 HR	9	8	17	11^{00}
Cost/Round		164	96	260	
Cost/Ton		2^{83}	1^{66}	4^{49}	

NOTE: Basis of estimate is one breast round of advance. One
round breaks 58 Tons. In one work place 2 men drill,
1 man mucks, loads holes, raises cribbing. Total
productivity is 58T x 2 rounds ÷ 3 MS = 38.7 T/MS;
116 T/Crew-shift

3. Cribbing labor included under stope labor.

4. Cost of cribbing - $500 per 1,000 board feet.

5. Jackleg labor cost - $\dfrac{260 \text{ ft.}}{40 \text{ ft/Hr.}}$ = 6.5 Hr.

6. Average explosives cost, including caps, $0.50 per ton.

A.4 BACKFILL SYSTEM

Data

1. Flow rate of slurry, 200 gallons per minute or 50 tons of dry solids per hour to fill one lift of 10-foot wide stope in three shifts of 6.5 hours' duration.

2. Stope fill crew to consist of three men.

3. Top 6-inch sand-cement mat of each fill lift requires 435 sacks of cement at $3.00 per sack.

4. Cost of sand assumed at $5.00 per cubic yard could be more or less dependent upon actual source.

5. Normal fill cycle will place 1,350 dry tons of sand.

Costs - Item 205

1. The makeup of the cost of labor and supplies to 1 cubic yard of fill at $7.94 per cubic yard is detailed in Table A3.

A.5 ORE RESERVES

Hypothetical ore reserves have been developed from the data available and the assumptions that the six veins chosen extend in depth to the level of their respective access adits. Idealized sections were drawn up, Figure A1 to A4, from which hypothetical volumes of veins were determined. From these, ore and mining reserves were assessed. These reserves are summarized in Table A4. A bulk density of 165 lb/ft^3 was assumed for the conversion of volumetric ore reserves.

TABLE A3

ESTIMATED COSTS OF SANDFILL

Cost Item	Units		Labor	Supplies	Total	Unit Cost
Filling Operations:						
U.G. Labor	9	MS	513		513	57
Pipe	1	LS		150	150	
Sealing Stope	9	MS	513	200	813	57
Cement	435	SK		1300	1300	3^{00}
Sand	1000	Cy		5000	5000	5^{00}
Cost/1000 Cy			1026	6650	7676	
Cost/Cy			1^{29}	6^{65}	7^{68}	

NOTE: Usual filling cycle would be: 300'x8'x11.6 ÷ 27 =
1000 Cy = 1350 Dry Tons Sand.

CHAIN PILLAR   1000 T
VEIN THICKNESS   1.5'

CHAIN PILLAR   6600 T
VEIN THICKNESS   3.8'

WEST
OREBODY

EAST
OREBODY

ELEVATION (ft.) ABOVE MEAN SEA LEVEL

7200

7100

7000

6900

6800

6750

6600

D.A.8

SOUTH FORK
AGENCY CREEK

LONGITUDINAL SECTION AT VEIN

0   100   200

SCALE IN FEET

BELT
SERIES

DIORITE DIKE

QUARTZ VEIN

BELT
SERIES

SECTION A-A'

figure A-1

IDEALIZED SECTIONS OF THE LUCKY STRIKE VEINS

CHAIN PILLAR   = 15,000 T
SILL PILLAR   = 8,000 T
TOTAL LOSS IN PILLARS = 23,000 T

A-A'   B-B'   C-C'

WEST OREBODY

LONGITUDINAL SECTION

BELT SERIES

QUARTZ VEIN

QUARTZ VEIN

BELT SERIES

SECTION A-A'

ELEVATION (ft) ABOVE MEAN SEA LEVEL

7800
7700
7600
7500
7400
7300

0   100   200
SCALE IN FEET

BELT SERIES

QUARTZ VEIN

QUARTZ VEIN

BELT SERIES

SECTION B-B'

BELT SERIES

QUARTZ VEIN

BELT SERIES

SECTION C-C'

ELEVATION (ft) ABOVE MEAN SEA LEVEL

7700
7600
7500
7400

figure A-2

IDEALIZED SECTIONS OF THE
LUCKY STRIKE AND TRAPPER VEINS

D.A. 10

CHAIN PILLAR   = 20,000 T
SILL PILLAR   = 21,000 T
TOTAL LOSS IN PILLARS = 41,000 T

CHAIN PILLAR   = 26,000 T
SILL PILLAR   = 18,000 T
TOTAL LOSS IN PILLARS = 44,000 T

A-A'   B-B'

SURFACE OUTCROP

EAST OREBODY

C-C'   D-D'

WEST OREBODY

GLORY
HOLE

LONGITUDINAL SECTION
LUCKY STRIKE No. 1

LONGITUDINAL SECTION
TRAPPER No. 1

ELEVATION (ft.) ABOVE MEAN SEA LEVEL

7600
7500
7400
7300

0   100   200
SCALE IN FEET

BELT
SERIES

QUARTZ VEIN

SECTION A-A'

BELT
SERIES

QUARTZ VEIN

SECTION B-B'

BELT
SERIES

QUARTZ VEIN

SECTION C-C'

BELT
SERIES

QTZ VEIN

QTZ VEIN

SECTION D-D'

ELEVATION (ft.) ABOVE MEAN SEA LEVEL

7600
7500
7400
7300

Figure A-3

IDEALIZED SECTIONS OF THE
LAST CHANCE VEIN

Figure A-4

CHAIN PILLAR = 7000 T
SILL PILLAR = 2000 T
TOTAL LOSS IN PILLARS = 9000 T

CHAIN PILLAR = 45,000 T
SILL PILLAR = 25,000 T
TOTAL LOSS IN PILLARS = 70,000 T

NARROW VEIN MINING

C-C'

A-A'

B-B'

ELEVATION (ft.) ABOVE MEAN SEA LEVEL

7800
7700
7600
7500

SCALE IN FEET
0   100   200

D.A. 11

A   DDH 3   A'
BELT SERIES
QUARTZ VEIN

B   DDH 4   B'
BELT SERIES
QUARTZ VEIN

C   DDH 4   C'
BELT SERIES   QUARTZ VEINS

ELEVATION (ft.) ABOVE MEAN SEA LEVEL

7800
7700
7600
7500
7400

Project No. _____ Reviewed _____ Date _____

TABLE A4
SUMMARY OF HYPOTHETICAL ORE AND MINING RESERVES

Vein	Sampled Grade %	Run of Mine Grade %	Ore Reserve Tons	Mining Reserve Tons	Mining Method
UPPER BLOCKS					
Wonder Lode West	0.19	0.15	28,000	19,000	R
Wonder Lode East	1.20	0.96	21,000	18,000	R
Lucky Strike No.1 East	0.02	0.02	298,000	231,000	CF
Lucky Strike No.1 West	0.25	0.23	254,000	208,000	CF
Trapper No.1 West	0.30	0.27	116,000	65,000	CF
Trapper No.1 East	1.45	1.31	13,000	8,000	CF
Last Chance A	0.47	0.42	329,000	233,000	CF
Last Chance B	0.47	0.38	40,000	28,000	R
	0.30	0.26	1,099,000	810,000	
LOWER BLOCKS					
Lucky Strike No.1 East	0.02	0.02	998,000	802,000	CF
Lucky Strike No.1 West	0.25	0.23	381,000	306,000	CF
Trapper No.1 West	0.30	0.27	818,000	659,000	CF
Trapper No.1 East	1.45	1.31	152,000	122,000	CF
Last Chance A	0.47	0.42	923,000	746,000	CF
Last Chance B	0.47	0.38	123,000	99,000	R
	0.32	0.28	3,395,000	2,734,000	

From Table A4 it is evident that 810,000 tons of hypothetical ore at a grade of 0.26 percent can be mined from the upper blocks and that 2,734,000 tons of hypothetical ore at a grade of 0.28 percent can be mined from the lower blocks. Overall recoveries from these veins are 74 percent and 81 percent for the upper and lower blocks, respectively.

A6   DEVELOPMENT AND CONSTRUCTION

The length of development and the number of chutes required to mine each vein were determined from the assumed geometry of the veins, as detailed in Table A5.

TABLE A5
DETERMINATION OF DEVELOPMENT AND CONSTRUCTION QUANTITIES

Type of Development/Construction	Algorithm
11'x10' Drift	Strike length ($\frac{\text{Vertical Depth}}{200}$ to next integer) x 1.25 = Length required.
Cut and Fill 6'x8' Raise	($\frac{\text{Strike length}}{300}$ to next integer) x Dip length x 1.10 = Length required.
Resue 5' dia. Raise	($\frac{\text{Strike length}}{150}$ to next integer) x Dip length x 1.10 = Length required.
Chutes (Cut and Fill)	($\frac{\text{Strike length}}{300}$ to next integer) x Number of levels. = Number required.
Chutes (Resue)	($\frac{\text{Strike length}}{150}$ to next integer) x Number of levels = Number required.

The development and construction quantities estimated for the various veins are summarized in Table A6.

TABLE A6
SUMMARY OF DEVELOPMENT AND CONSTRUCTION QUANTITIES

	Length of Development (ft.)			
Vein	Drift	Raise 6' x 8'	Raise 6' x 8'	No. of Chutes
UPPER BLOCKS	11'x10'			
Wonder Lode	3,000	--	400	7
Trapper No.1 East & West				
Lucky Strike No.1 East	2,075	800	400	5
Lucky Strike West	3,850	1,600	500	6
Last Chance Cut and Fill	1,900	800	350	3
Last Chance Resue	1,000	--	900	2
Sub-Totals	11,825	3,200	4,950	23
LOWER BLOCKS	10'x10'			
Lucky Strike No.1 East	3,100	1,900	--	8
Lucky Strike No.1 West	3,500	3,600	--	12
Trapper No.1 West	3,000	1,600	--	8
Trapper No.1 East	300	800	--	4
Last Chance Cut and Fill	5,400	3,700	--	15
Last Chance Resue	2,500	--	6,700	30
Sub-Totals	17,800	11,600	6,700	77

Data

11'x10' Trackless Drift

1. Productivity of manpower - 3.5 feet per manshift, three-man crews.

2. Length of round 10', actual advance 9'.

3. Explosives used, 5 lb. per cubic yard of rock, dynamite or slurry.

4. Two-boom trackless jumbo with 10' feed, utilized for drilling.

5. Development dirt lashed by a 2-cubic-yard capacity LHD with ejector bucket. Dirt hauled by 10-ton truck with push plate dumping facility.

10'x10' Track Drift

6. Productivity of manpower - 2 ft. per man-shift, 3 man crew.

7. Length of round 10', actual advance 9'.

8. Explosives used 5 lb. per cubic yard of rock, dynamite, or slurry.

9. Two-boom rail jumbo with 10' feed, utilized for drilling.

10. Air-powered overshot mucker with conveyor belt used for lashing development dirt into granby cars hauled by trolley locomotive.

6'x8' Stulled Raise

11. Productivity of manpower - 2.75 ft. per man-shift, 2 man crew.

12. Length of round 6', actual advance 5.5'.

13. Explosives used 5.28 lb. per cubic yard of rock, dynamite or slurry.

14. Round drilled by hand-held stopers.

5' dia. Bored Raise

15. Productivity of manpower - 6.25 ft. per man-shift, 2 man crew.

16. Set up time assumed at three days and drilling time for 200' raise estimated at five days.

Construction

17. Productivity of manpower assumed for chute construction 20 manshifts per chute, including 10' of 4'x6' raise.

18. Portals costed on basis of the additional labor and materials required over and above the standard drifting cost.

19. Dozer time for construction of portals assessed at two shifts for a 50' long portal. Labor for portal construction estimated at 30 man-shifts.

20. Assessment of support necessary for portals is a 6-inch lining of shotcrete over 50: i.e., 28.7 cubic yards per portal.

Costs

1. Estimates of the labor and materials needed for the various types of development are given in Table A7 and details of construction costs are enumerated in Table A8.

TABLE A7
ESTIMATED COSTS OF DEVELOPMENT

Items 220, 230, 240 and 241	Units		COSTS ($)			
			Labor	Supplies	Total	Unit
11'x10' Trackless Drift (220)						
Miners	2.75	MS	231.00		231.00	90.00
Drill steels & bits	400.00	LF		60.00	60.00	0.15
Explosives	203.00	LB		122.00	122.00	0.60
6" Air line	9.00	LF		117.00	117.00	13.00
2" Water line	9.00	LF		22.00	22.00	2.45
Jumbo	5.00	HR	36.00	29.00	65.00	13.00
LHD	3.00	HR	9.00	37.00	46.00	15.25
10T Truck	2.00	HR	5.00	13.00	18.00	8.65

Cost per round $681.00

Cost per foot $75.00

10'x10' Track Drift (230)						
Miners	4.50	MS	405.00		405.00	90.00
Drill steels & bits	360.00	LF		54.00	54.00	0.15
Explosives	185.00	LB		111.00	111.00	0.60
6" Air line	9.00	LF		117.00	117.00	13.00
2" Water line	9.00	LF		22.00	22.00	2.45
Rail	18.00	LB		108.00	108.00	6.00
Ties	3.00	EA		24.00	24.00	8.00
Track Miscellaneous	9.00	LF		9.00	9.00	1.00
Jumbo	4.50	HR	27.00	20.00	47.00	10.50
Loader	4.00	HR	24.00	20.00	44.00	11.00
Trolley Locomotive	6.00	HR	38.00	33.00	71.00	11.75
Granby cars	20.00	HR	16.00	16.00	32.00	1.60

Cost per round $1,044.00

Cost per foot $116.00

6'x8' Stulled Raise (241)						
Miners	2.00	MS	180.00		180.00	90.00
Drill steels & bits	200.00	LF		20.00	20.00	0.10
Explosives	50.00	LB		30.00	80.00	0.60
4" Air line	5.50	LF		33.00	33.00	6.00
1" Water line	5.50	LF		6.00	6.00	1.00
Timber	200.00	BF		40.00	40.00	0.20
Ladders	11.00	LF		33.00	33.00	3.00
Stoper	7.00	HR	21.00	24.00	45.00	6.50

Cost per round $387.00

Cost per foot $70.00

5' dia. Bored Raise (240)						
Miners	32.00	MS	2,880.00		2,880.00	90.00
Cutters	3,927.00	CuFt		5,890.00	5,890.00	1.50
Raise Borer	80.00	HR	1,808.00	2,140.00	3,948.00	49.35

Cost per round $12,718.00

Cost per foot $64.00

TABLE A8
ESTIMATED COSTS OF CONSTRUCTION

ITEMS 223, 250		COSTS ($)			
	UNITS	LABOR	SUPPLIES	TOTAL	UNIT
Chutes (250)					
Chute Raise	10.00 LF	200.00	90.00	290.00	29.00
Labor	20.00 MS	1,800.00		1,800.00	90.00
Chute Timber			454.00	454.00	
Air Cylinder			1,000.00	1,000.00	1,000.00
Rollers	8.00 EA			160.00	20.00
Used Rail	100.00 LF			400.00	4.00
Misc. Hardware & Pipe Fittings			500.00		
Misc. Timber	200.00 BF			40.00	0.20
				$4,644.00	

Cost per chute $4,644.00

Portals (223)					
Portals (223)					
Shotcrete	28.70 CY	795.00	640.00	1,435.00	50.00
Miners	30.00 MS	2,700.00		2,700.00	90.00
Dozer	16.00 HR	310.00	268.00	578.00	36.15
				$4,713.00	

Cost per portal $4,713.00

A.7   OPERATING COSTS

Ventilation

1. Each working place must be supplied with air at a velocity of 100 f.p.m.

   On this basis, cut and fill stopes require 10,000 cfm per stope; resue stopes 6,000 cfm each; drifts 11,000 cfm each.

2. A brief summary of the ventilation requirements for the mine are enumerated in Table A9.

TABLE A9
SUMMARY OF VENTILATION REQUIREMENTS

Working Place	Total Air Quantity c.f.m.	Fans Required	Total Power
Cut and Fill stopes (8)	80.000	8 x 20 H.P.	160
Resue stopes (4)	24,000	4 x 10 H.P.	80
Drifts and raises (6)	66,000	6 x 20 H.P.	120
Workshops, etc.	10,000	4 x 10 H.P.	40
Miscellaneous	25,000	5 x 10 H.P.	50
	205,000		450

3. Leakage assessed at 30 percent of total air quantity.

4. Ventilation ducting purchased as capital equipment.

Tramming

   Trackless Haulage

1. Requirements based on: 4 m.p.h. traveling speed; 4 minute fixed journey time; 10-ton load; 50 minute hour. Thus, 1,500 feet hauling productivity is approximately 40 tons per hour, or 260 tons per shift.

2. Three 10-ton trucks are needed for the Wonder, Last Chance and Trapper-Lucky Strike veins owing to the distance between portals.

3. Trucks used for supply when not hauling.

4. Productivity would, therefore, be at a production rate of 500 tons per shift, 167 tons per machine shift.

5. Labor cost $316 per shift, supplies $123 per shift, thus cost per 500 tons is $439 or $0.88 per ton hauled.

Rail Haulage

1. The decision to utilize rail haulage on the main 6600 level was based on the large tonnages and long haul distances involved (Table A10).

TABLE A10
HAUL DISTANCES AND TONNAGE OF LOWER BLOCKS

Vein	Mining Reserve (tons)	Haul Distances (feet) Average	Longest
Last Chance	369,000	1,000	1,500
Lucky Strike E	298,000	1,250	1,500
Lucky Strike W	1,980,000	2,100	2,400
Trapper	129,000	500	950
Wonder	49,000	400	1,100

2. A factor of 174 minutes travel time per shift is estimated for a speed of 528 fpm to haul 500 tons, i.e. seven trips per shift with 71 ton loads.

3. Ten-ton loco required to haul 10 7-ton granbies.

4. Breakdown of rail haulage costs are listed in Table A11.

TABLE A11

ESTIMATED COSTS OF RAIL HAULAGE

ITEM 270	Units	COSTS ($)			
		Labor	Supplies	Total	Unit
Motor Man	1.00 MS	90.00		90.00	90.00
Chute Puller	1.00 MS	90.00		90.00	90.00
Trolley Loco	6.50 HR	41.00	36.00	77.00	11.75
Granby Cars	45.50 HR	36.00	36.00	72.00	1.60
	Cost per 500 tons			329.00	

Cost per ton $0.66

Compressor Plant

1. The 10,000 cfm quantity of compressed air necessary to operate the air-powered equipment in the mine was based on the estimates of consumption for the various activities. These consumptions are summarized in Table A12.

TABLE A12

SUMMARY OF COMPRESSED AIR REQUIREMENTS

Activity	Compressed Air c.f.m.
Resue stopes: 4x150 cfm	600
Cut and Fill Stopes:	
4 muckers x 280 cfm	1,120
4 crews x 2 drills x 150 cfm	1,200
Drifts: 4 crews x 2 drills x 500 cfm	4,000
Raise Borer	850
Diamond Drill	750
Miscellaneous: tuggers, chutes, small tools	1,000
	9,820 cfm

2. Diversity factor for 30 operating pieces of equipment

    $= \dfrac{23.5}{30} \quad = 0.78.$

    Altitude multiplier = 1.22.

    > Estimated consumption = 9,820 x 1.22 x 0.78 cfm
    >   = 9,285 cfm
    > Rounded up to 10,000 cfm

3. Power requirements for compressor plant can be estimated from a factor of 20 brake horsepower per 100 cfm. Thus 2,000 HP synchronous motor is needed to produce 10,000 cfm.

4. Power cost of motor can be calculated as follows:

    > 2,000 x 0.746 x $0.025 x 1/1 = $37.30/HR.
    > Cost per annum = $149,200

5. Power cost of motor represents 65% of the running costs of a compressor plant compared to 20% for operating charges and 15% for fixed charges and repairs. Thus:

Power cost 65%	$149,200/YR.
Operating charges 20%	$ 46,000/YR.
Fixed charges and repairs 15%	$ 34,000/YR.
Annual operating cost	$229,200/YR.

Electricity Supply

1. The annual power costs per annum at a production rate of 250,000 tons per year are listed in Table A13.

TABLE A13

ANNUAL POWER COSTS

Item	Annual Costs $
Ventilation fans (0.80 power factor)	92,000
Tramming	6,400
Lighting Underground	21,900
Slushers & heaters (0.80 power factor)	20,000
Compressor plant	149,200
Lighting surface, heaters, etc.	20,000
Sub Total	309,500
Contingencies	30,000
	340,000 per year

Transportation

1. The provision of transport facilities for the staff and
    personnel is detailed in Table A.14 together with the
    assumed running costs.

TABLE A14

OPERATING COSTS OF PERSONNEL AND SUPPLIES TRANSPORT

Personnel	Vehicle	Estimated Miles Per Year	Cost $ Per Mile	Cost $ Per Year
Mine Manager	Saloon	52,000	0.21	10,920.00
Superintendent	Pick-up	22,500	0.21	4,725.00
Foreman	Pick-up	22,500	0.21	4,725.00
Foreman	Pick-up	16,250	0.21	3,410.00
Personnel (from Dillon)	Bus 1	45,000	0.50	22,500.00
Personnel (from Dillon)	Bus 2	45,000	0.50	22,500.00
Personnel (from Salmon)	Bus 3	30,000	0.50	15,000.00
Supplies	5T Truck	20,800	0.50	10,400.00
				$94,180.00 Per Year

D.A.23

2. Mileages for the buses were calculated on the assumption of a 250-day year and two journeys per day.

Hourly Equipment Costs

1. The basis for the hourly equipment costs charged to the various cost centers is outlined in Table A14.

TABLE A14
HOURLY EQUIPMENT COSTS

ITEM	COSTS ($)/HOUR					
	Repair Supplies	Filter, Lube, Tires,	Repair Labor	Total Supplies	Total Labor	Total Cost
LHD 2 Cubic Yard	3.00	9.25	3.00	12.25	3.00	15.25
10-Ton Haul Truck	2.35	3.95	2.35	6.30	2.35	8.65
Trackless Jumbo	4.40	1.40	7.20	5.80	7.20	13.00
Jacklegs & Stopers	3.00	0.50	3.00	3.50	3.00	6.50
Raise Borer	21.10	5.65	22.60	26.75	22.60	49.35
1.75-Ton Loader	4.15	1.00	5.85	5.15	5.85	11.00
Granby Car 135 cf	0.75	0.05	0.80	0.80	0.80	1.60
Trolley Loco 11 Ton	5.10	0.40	6.25	5.50	6.25	11.75
Slusher 2 Drum 15 HP	2.50	1.00	2.50	3.50	2.50	6.00
D8 Dozer	10.00	6.75	19.40*	16.75	19.40	36.15
Rail Jumbo	3.50	1.00	6.00	4.50	6.00	10.50
Mucker	4.00	1.00	6.00	5.00	6.00	11.00
Boss Buggy U/G	2.50	4.00	2.50	6.50	2.50	9.00
Diamond Drill	2.00	3.00	2.00	5.00	2.00	7.00

*Includes Operating ($11.60/HR) and Service ($2.80/HR) Labor.

Indirect Operating Costs

The indirect operating costs for the life of the mine amount to $19,726,159 (Tables A16 and A17). The indirect costs for the first seven years of production are greater than the latter half of the mine life since development and exploration activities are carried out during the initial years of production.

A8   CAPITAL COSTS

1.   The estimate of expenditure for capital equipment is based on approximate costs, free on rail, given by some of the major mining equipment manufacturers.

2.   Building costs are estimates made on the basis of $15 per square foot of the approximate area required.

3.   The mine life is assumed to be 14 years for capital items normally written off over the working life of a mine.

4.   Equipment ordinarily due for replacement within the 13th and 14th years of production have not been replaced.

APPENDIX E

PALMER, MICHIGAN MODEL MINE/MILL

This appendix contains a report completed for PNL by a consulting agency.

Golder Associates

CONSULTING MINING AND GEOTECHNICAL ENGINEERS

REPORT TO
BATTELLE-PACIFIC NORTHWEST LABORATORIES
FOR
MINING AND MILLING A THORIUM-BEARING QUARTZITE
AT PALMER, MICHIGAN
BY OPEN PIT METHODS

Distribution:

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GOLDER ASSOCIATES, INC., 10628 NE 38th Place, Kirkland (Seattle), Washington 98033, U.S.A. — Phone (206) 827-0777, Telex 32-1014

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1.0   INTRODUCTION

This report examines the estimated costs of mining and concentrating the thorium-bearing mineral monazite from a rock formation known as the Goodrich Quartzite, which occurs under and around the town of Palmer, Michigan.

The amount of monazite contained in the Goodrich Quartzite and the amount of thorium contained in the monazite have been inferred only from very scanty data and from tests made and reported in the mid 1950's. These inferred percentage amounts have been used as a basis for this study. As the Goodrich Formation is near the surface, open pit mining methods were considered.

The mining and milling process would end with the production of a concentrate largely containing monazite which would be loaded into rail cars for shipment to a refinery. The overall process would start with the advance stripping of waste glacial till and rock to expose the thorium-bearing portions of the Goodrich Formation. This stripping of waste and the subsequent mining of ore would be done by large electric shovels loading into mining trucks. The Goodrich Quartzite is a hard rock and would have to be broken prior to loading by drilling and blasting. The waste would be hauled to a waste dump and the ore to a concentrator, where the monazite component of the ore would be separated. The remaining gangue would be slurried to a tailings storage area.

In order to obtain a more complete estimate of the costs of this process and the work required to bring a mine into production, an exploration, pre-mine development, construction, and operational schedule was formulated. The costs of each of

these components were estimated and then broken into annual sums.

The schedule was divided into pre-revenue and revenue periods (revenue is assumed to commence when the mill is onstream producing concentrate). Ten years of pre-revenue activity were allocated to explore and develop the Goodrich Quartzite, and to construct the concentrator and mine buildings. The annual production rate from the mine during the revenue years was set to give it approximately twenty years of operational life.

Because of the general level of uncertainty that is associated with the grade, grade distribution, and concentrating characteristics of this deposit, order-of-magnitude cost estimates have been used. These are based on current industrial experience in processes most relevant to those envisioned as being applicable to the recovery of thorium in the Goodrich Quartzite.

2.0 GENERAL DESCRIPTION OF THE PALMER AREA

2.1 Location

The Palmer area is in the east-central portion of Marquette County, Michigan, which occupies a part of the northern penninsula of Michigan (see Figure 1). The town of Palmer is located about 10 miles southeast of the iron-ore shipping port of Marquette. The area is served by a network of good highways and railroads. The population of Palmer is approximately 800 according to Marquette County officials.

The general area of interest is approximately 4 miles in length and .75 miles in width. It occupies parts of sections 27 through 34, T47N, R26W, and includes the town of Palmer. The Palmer area is part of a major iron-mining region. Three miles north of the area is Negaunce, site of the first discovery of iron ore in the Lake Superior region. About 1 mile northwest of the Palmer townsite is the Empire iron mine, which currently produces 5.2 million tons per year (tpy) of iron pellets. A 250 million dollar expansion program is underway to expand this to 8 million tpy by 1980.

2.2 Geography

The Palmer area is characterized by relatively rugged, knobby topography, and is covered by a dense second-growth forest. Elevations range from 1,160 feet above sea level in the southeast to a maximum of 1,780 feet about 1.5 miles northwest of Palmer. The most prominent topographic reliefs are the Kona Hills and Wewe Hills to the northeast, and a small belt of hills north and south of Palmer. To the east is an extensive sand plain.

(AFTER VICKERS, 1956, p. 173)

The area is entirely within the Lake Michigan drainage basin, and the localized drainages have a distinctive southeastward configuration. The largest stream is the East Branch of the Escanaba River, which forms from the confluence of Warner and Schweitzer Creeks and flows southeastward to Lake Michigan. The Goose Lake system drains the areas to the northeast and east. Small swamps occur in widely different elevations in much of the area.

2.3 Climate

The nearest climatological stations which are operated and maintained by the National Oceanic and Atmospheric Administration, U.S. Environmental Data Service, are located in downtown Marquette (latitude 46°34' north, longitude 87°24' west elevation 677 feet above sea level), and about 8 miles south at the Marquette Airport (latitude 46°32' north, longitude 87°34' west, elevation 1,415 feet above sea level). Detailed tables giving normals, means, and extremes of the various weather data collected by these stations are available from NOAA.

The following excerpt from a 1963 summary gives the salient climatolical features of the region:

"Based on the record to 1963 for the downtown Marquette station, the average data of the last occurrence of freezing temperatures in spring is May 14 and the average data of the first freezing temperature in autumn is October 20. The length of the growing season ranges between 134 and 182 days and averages 159 days annually. The warmest year on record was 1931 with an average annual temperature of 46.8°F, and the coldest year was 1885 with

an average temperature of 36.6°F. Precipitation is rather
evenly distributed throughout the year, there being no
pronounced wet or dry periods. Total Precipitation for
the driest year on record, 1925, was 19.68 inches and for
the wettest year, 1881, 42.70 inches. Snowfall is heavy,
exceeding 100 inches in 7 out of every 10 winter seasons.
The greatest snowfall on record for any winter was 189.1
inches during the 1890-91 season, followed closely by the
winter of 1959-60 with 188.1 inches and that of 1949-50
with 188.0 inches. Least snowfall for any seasn was 53.4
inches in 1940-41. The greatest depth of snow on the
ground was 48.6 inches and occurred on Januay 2, 1911."
(U.S. Department of Commerce, Weather Bureau, 1963).

3.0 GEOLOGY

3.1 General - Palmer Area

The geology of the area surrounding Palmer is thoroughly
covered in U.S. Geological Survey Professinal Paper 769 on
"Bedrock Geology and Ore Deposits of the Palmer Quadrangle,
Marquette County, Michigan" by Jacob E. Gair, 1975. The ore
deposits mentioned in the title are iron deposits; the
thorium-bearing Goodrich Quartzite is of minor importance within
the intent of that publication.

3.2 The Goodrich Quartzite

3.2.1 General

The Goodrich Quartzite in the Palmer area is composed of a
locally-developed basal conglomerate which grades upward into
interbedded quartzites and conglomerates. The lithology of the
basal conglomerate varies with the character of the subjacent
Negaunne Formation. The basal portion of the Goodrich is
composed almost entirely of boulders and pebbles of granite.
Above the basal unit, the Goodrich Quartzite consists of
alternating beds of coarse quartzite and pebble conglomerate.
Thicknesses of the observed conglomerate beds or lenses range
from approximately 2 inches to 2 feet. The separating quartzite
beds of coarse sand range from 1 to 30 feet in thickness. The
observed relationships in this area are limited to outcrops
which represent only the lower 200 feet of the formation.

The maximum thickness of the Goodrich Quartzite has not
been determined. Three diamond drill holes penetrated
approximately 1,100 feet of the Goodrich, but no younger
formations were encountered (Vickers 1956).

The monazite concentrations are associated with the pebble conglomerates. Only the upper 700 feet of the three diamond drill holes were gamma-ray logged. The lower portion of these holes exceeded the length of cable available for the gamma ray probe. These logs show a general increase in radioactivity in the upper and middle portions of the formation. This led Vickers (1956) and Gair (1975) to assume that the percentage of conglomerate beds proportionally increased in the upper part of the formation. For the purposes of this theoretical study, that untested assumption has been retained.

The main structural features in the area are the eastward-striking Palmer fault, several faults of smaller displacement, and folds. The Palmer fault separates the Palmer area from the main portion of the Marquette synclinorium. The Palmer area occupies the south limb of the synclinorium.

3.2.2   Shape and Thorium Content

The shape, or geometry, of the Goodrich Quartzite has been taken from the geologic map produced by Gair (1975) and must be considered to be approximate only, as sufficient drilling has not been done to confirm the precise locations of several of the boundaries.

The grade, or thorium content, has been evaluated by Gair (1975, pp. 144-146), and is based on a curve he developed relating the thorium content of samples from the Goodrich (measured by gamma ray spectrometer) and the radioactivity (in microengens per hour) observed at the sample sites. The results of his estimates are tabulated on Plate 7 of his paper, in which

the Goodrich is divided into three areas: 1) Central (which is divided into the upper and lower), 2) Isabella Syncline, and 3) Old Volunteer Mine. Examination of the thorium grades estimated for these areas quickly shows that the upper part of the Central Area is the portion of the Goodrich that contains the highest grade. Thus, the Central Area was chosen as the "conceptual orebody" for this study. In estimating the grade and acertaining the shape of this portion of the Goodrich, Gair leans heavily on work done by Vickers (1956).

We are uncertain as to whether Gair uses thorium assays taken from the core examined by Vickers, or whether he uses the radiometric logs Vickers obtained and published when be probed the holes with a gieger counter. In any event, the grade of 1.06 pounds of thorium per ton was used as the grade basis of the upper portion of the central area in this study. Monazite was assumed to be evenly distributed throughout this part of the deposit for the purposes of mine and concentrator planning. It must be noted that, in fact, this is not the case, as the conglomeratic portions of the upper Central Goodrich Quartzite contain higher radiometric values (hence presumably more monazite) than the quartzite portions. Recognizing this weakness, but also noting that the quartzite portions contain enough radioactivity to make them at least low-grade ore (if Gair's correlation is correct), it is possible that the entire upper portion of the Central Area could be high enough to cover milling costs. Hence, this simplifying assumption is considered reasonable, given the very limited data available.

Vickers had two assays made to establish the thoria (ThO_2) content of the monazite found in the Goodrich. These suggest that the thoria content of this mineral is 8.2 percent

(also see Overstreet, 1967, p. 167). Accepting the 1.06 pounds Th per ton figure estimated by Gair (for the upper part of the Central Area), the following assumptions were drawn and used in this study: 1) The upper Central Area of the Goodrich Quartzite contains 1.06 pounds per ton of thorium (Th) which is equivalent to 1.21 pounds of thoria (ThO_2); and 2) As the thoria is 8.2 percent of the monazite, the average monazite content must be 14.7 pounds per ton.

3.2.3 Upper Part of the Central Area

3.2.3.1 Mining Geology and Pit Design

As discussed above, the upper part of the Central Area of the Goodrich Quartzite was selected for conceptual mine purposes. Its shape is based on the "Bedrock Geologic Map and Sections of the Palmer Quadrangle, Marquette County, Michigan," Plate 1 of Gair's 1975 report, and the grade is based on Gair's evaluation of Vicker's work done in 1956.

In order to obtain an appreciation of the total tonnages involved (hence, the potential production rates) and the final shape of an open pit, the following map work was undertaken. Based on the cross sections, the areal extent of the Goodrich Central Area, and the structure interpreted by Gair, a contour map was constructed at a scale of 1 inch equals 2,000 feet, with a contour interval of 50 feet. The contour map (Figure 2) represents the shape of the conceptual orebody. This conceptual orebody has an assumed average grade of 1.06 pounds of Thorium per ton, which is contained in 14.7 pounds of monazite per ton.

CONTOUR MAP OF CONCEPTUAL OREBODY
(OVER BEDROCK GEOLOGY)

Figure 2

Three pits were designed to remove increasingly larger portions of this assumed orebody. The first has a western edge just to the east of the town of Palmer, avoiding the need to purchase and remove that town. Except for the overlying till, which is assumed to be uniformly 30 feet thick over all of the pits, the pit envelops only ore shown on the contour map of the orebody discussed above.

The second pit has as its boundaries the total areal extent of the upper Central Goodrich, thus, it covers most of the town of Palmer.

The third pit is the same as the second, with the exception of the north wall, which intersects the upper Goodrich halfway between its bedrock surface and its contact with lower Goodrich along the Palmer fault. This permits the excavation of a considerably larger amount of the upper Central Area "orebody" but, of course, also includes more waste rock. After discussing these options with Battelle Northwest, it was decided to use this as the basis for the costing exercise.

The various pits are shown as Figures 3, 4 and 5. Figure 6 shows the bedrock geology and cross sections immediately around the Central Area of the Goodrich Quartzite. All of the pits have 45 degree walls, a reasonable assumption in competent rock such as quartzite. The pits were measured in volume and the specific gravity of all rock was assumed to be 2.7. The amount of ore and waste contained in each pit is summarized in Table 1. The cross sections showing the various pits projected onto the Upper Goodrich are displayed in Figure 7.

PIT 2
(OVER BEDROCK GEOLOGY)

Figure 4

E.14

PIT 3
(OVER BEDROCK GEOLOGY)

Figure 5

E.15

GOODRICH QUARTZITE
CENTRAL AREA

☐ UPPER

▨ LOWER

– – – – FAULTS

DRILLHOLES
(DETAILS ON REFERENCE BELOW)

–·–·– PIT OUTLINES

0   1000   2000
SCALE IN FEET

(AFTER GAIR, 1975, PLATE I)

E.16

Figure 7

TABLE 1

ORE & WASTE IN PITS 1, 2 and 3

PIT	ORE (TONS)	WASTE (TONS)	SR*
1	49,000,000	8,300,000	0.17
2	111,000,000	16,000,000	0.14
3	205,000,000	55,000,000	0.27

*Tons Ore
Tons Waste

The largest pit, which was selected for further study contains 205 million tons of rock containing an assumed 14.7 pounds of monazite per ton, with an assumed 8.2 percent ThO_2 content. The pit also contains 55 million tons of waste rock and till. Using a 20-year production life, 10 million tons of ore could be produced each year from the selected pit. In addition, about three million tons per year of waste rock and till would have to be moved.

3.2.3.2 Concentrator Recoverability

To our knowledge, no tests have been made on the Goodrich to determine the overall recoverability of the monazite minerals (the percentage of monazite recovered in the concentrate and an well the percentage of gangue or waste minerals reporting to that concentrate). Vickers (1956) reported that heavier minerals, including monazite, could be separated after crushing and grinding, but did not attempt to quantify the percentage recoverability. Borrowman and Rosenbaum (1962, p.2-3) indicate the successful laboratory separation of monazite from the Deadwood Conglomerate found at Bald Mountain, Wyoming. The overall recovery of monazite was 61 percent by weight. Our interpretation of these results is shown in Figure 8.

Assuming that the Goodrich behaves in much the same manner as the Deadwood (an argument can be advanced that they both may have at least similar monazite grain size distributions), then a materials balance would result as indicated on Figure 9.

Lacking other data, the assumption that a 61 percent overall recovery through the concentrator was applied to the Goodrich in this study. The concentrator would require the following elements:

DEADWOOD CONGLOMERATE
BALD MOUNTAIN, WYOMING:
MATERIALS BALANCE LABORATORY TEST

(Based on: Borrowman S.R and Rosenbaum, J.B. 1962.
Recovery of Thorium from a Wyoming Ore. Bu Mines RI 5917.)

Figure 8

MILL
FEED

(0.38 ThO₂)
4.32 Monazite
2000
1995.68 Other

CRUSHING and
GRINDING to 20 Mesh
& GRAVITY TABLING

15

(0.26 ThO₂)
2.95 Monazite
15
12.05 Other

DRYING &
MAGNETIC
SEPARATION

4.2

(0.24 ThO₂)
2.70 Monazite
1.50 Other

CONCENTRATE

1985

(0.12 ThO₂)
1.37 Monazite
1983.63 Other

10.8

(0.02 ThO₂)
0.25 Monazite
10.55 Other

1985   1995.8   10.8

(0.14 ThO₂)
1.62 Monazite
1994.18 Other

4.

TAILS

E.19

READ ALL VALUES AS POUNDS

GOODRICH QUARTZITE
PALMER, MICHIGAN:
MATERIALS BALANCE FOR A CONCEPTUAL MILL

(Based on: Borrowman, S.R. and Rosenbaum, J.B. 1962.
Recovery of Thorium from a Wyoming Ore. Bu Mines RI 5917.)

MILL FEED

(1.06 Th
1.21 ThO₂) → $(1.06\ Th, 1.21\ ThO_2)$
14.76 Monazite
1985.24 Other
2000

CRUSHING & GRINDING
& GRAVITY SEPARATION

51.76

$(0.83\ ThO_2)$
10.18 Monazite
41.58 Other
51.76

1948.24

$(0.38\ ThO_2)$
4.58 Monazite
1943.66 Other
1948.24

DRYING &
HEAVY MINERAL
SEPARATION

14.00

$(0.74\ ThO_2)$
9.00 Monazite
5.00 Other

CONCENTRATE

37.76

$(0.09\ ThO_2)$
1.18 Monazite
36.58
37.76

1986.00

$(.47\ ThO_2)$
5.76 Monazite
1980.24

TAILS

READ ALL VALUES AS POUNDS

E.20

figure 9

1. A crushing and grinding circuit which would reduce the run of mine ore to optimum size (probably -20 mesh).
2. A gravity section (probably spirals) which separates the heavy minerals from the lighter gangue minerals.
3. A heavy mineral section which would separate the heavy minerals (i.e., rutile, ilmenite, and magnetite) from monazite.
4. A tailings disposal section which would transport and place the mill tailings to the tailings pile.

4.0 DEVELOPMENT SCHEDULE

4.1 General

In order to examine and estimate costs for the many and diverse tasks that need to be completed before the mine and mill could successfully be brought onstream, a development schedule was prepared and is attached as Figure 10. The schedule is divided into years, with a zero point designated as the time at which the mill would commence production of monazite concentrate. The sales of this concentrate would presumably produce revenue. For this reason, the years after the zero point are designated "revenue" and the years before "pre-revenue". Eleven pre-revenue years requiring large expenditures of capital are thought to be necessary to bring the mine into production in an orderly way.

Four different activities or phases have been identified: exploration, pre-mine development, construction, and operations. The purpose and scope of each of these phases is discussed in more detail below and costed in Section 5.0.

4.2 Exploration Phase

4.2.1 Purpose

The shape and distribution of the monazite, within the Goodrich the thorium content of the monazite, and the physical characteristics of the ore that will determine the recovery of the monazite in a concentrating plant, are not well understood. The assumptions made for the purposes of this study would have to be verified before any further funds could prudently be ventured. The U.S. Geological Survey classifies the Goodrich Quartzite as an identified resource which, in terms of certainty,

PALMER THORIUM MINE : MINE & MILL EXPLORATION, DEVELOPMENT, CONSTRUCTION & OPERATIONAL SCHEDULE   FIGURE 10

is far from being a possible, much less proven, orebody.

For these reasons, mineral exploration will have to precede all the other phases. Assuming these results are favorable, the project could then proceed.

4.2.2 Details

This phase has been divided into two important subtasks. The first of these is land acquisition. During this subtask, enough mineral rights with overlying surface rights would be tied up, probably by means of options that could be exercised later, to ensure the success of the operation should exploration results prove favorable. Also required would be landowners' permission to allow access for drilling and other necessary tests during the drilling portions of the exploration phase.

The second subtask, exploration geology, has been further subdivided into two subphases. During the first subphase, several holes (15 are thought to be sufficient) would be cored through the Goodrich. These holes would test the concept of the resource, both from a geometrical and grade/tenor standpoint, and also develop enough physical data on the mineralized zone to enable judgements to be made with some assurance about recoverability of the monazite mineral in a concentrator. At the end of first subphase drilling, the deposit would be interpreted geologically and a prefeasibility study would be done. If this study were favorable, second-subphase drilling would be allowed to continue (having been started before the study was completed). If the prefeasibility study is unfavorable, the whole project, including the land options, could be dropped.

The second subphase of drilling would require about 85 holes. Assuming that a reliable correlation can be developed between thorium content and amplitude of the radiometric log, only 22 holes would require complete coring. The other 63 could be drilled and logged radiometrically, which would be less expensive than coring.

The 100 holes from subphases one and two would now be reinterpreted and another prefeasibility study made. If the results were favorable, the deposit would enter the pre-mine development phase as a largely proven orebody. Through both drilling phases, enough metallurgical tests (grain size, grindability, and recoverability analyses) would be necessary to assure the recovery of the monazite mineral in a concentrator. Three years is estimated to be required to complete the exploration phase.

4.3 PRE-MINE DEVELOPMENT PHASE

4.3.1 Purpose

The purpose of this phase is to: 1) continue the land acquisition program; 2) completely delinate and determine grade and recoverability distributions within the orebody; 3) perform a first-order technical and economic feasibility study; and 4) obtain the necessary mining permits to allow construction, mining and mill operations to proceed.

4.3.2 Details

Land acquisition would continue, with surface and mineral rights being purchased or leased as necessary. The mine geology subtask would consist of sinking another 200 holes to completely delineate the orebody and determine the grade distribution of the monazite mineral and its recoverability by milling. In

addition, a large bulk sampling program would be required to provide ore to a pilot mill. The purpose of the pilot mill is to provide detailed mill design parameters. This program would obtain ore from underground workings by sinking a shaft in a "representive" spot in the orebody, and by driving drifts at various vertical intervals down the shaft.

As certain portions of this data become available, feasibility studies would commence. These studies would have as an overall purpose the mine design and costs, the mill design and costs, and an overall project cost determination, as well as a financial analysis.

Assuming favorable results, necessary permits would be sought. These would probably require environmental impact studies, followed by submission of plans, hearings and, hopefully, the issuance of permits.

Four and one-half years are thought to be necessary to complete the pre-mine development phase in an orderly fashion.

4.4 Construction Phase

4.4.1 Purpose

During this period, the stationary plant required for the project will be constructed. This will consist of a concentrator, mine service buildings, power lines, access roads, and all other items that are required to make production possible.

4.4.2 Details

In this study, all construction, including final design, procurement and construction management, will be assumed to be handled by an engineering construction firm. Their services will not extend to pre-mine stripping, a point which will be covered below.

The construction phase has been assumed to require three years to complete.

4.5 Operational Phase

4.5.1 Purpose

The purpose of mine and mill operations is to perform the operations necessary to produce a salable concentrate throughout the life of the project.

4.5.2 Details

4.5.2.1 Mine Operations

An excavation schedule has been formulated (Table 2) which indicates the annual tonanges of ore and waste that are estimated to be required to maintain a constant mill feed to the concentrator.

Normally, such a schedule is based on a detailed mining excavation plan which shows, bench by bench, pit expansion by pit expansion, the flow of ore and waste throughout the life of the mine. Such a detailed study is not warranted, and would normally not be made until the feasibility stage of the pre-mine development phase. The schedule presented herein is simply

TABLE 2

ANNUAL EXCAVATION SCHEDULE – PALMER THORIUM MINE

YEAR	WASTE	ORE	ORE & WASTE
	(Millions of Tons)		
-2	3		3
-1	3		3
1	3	6	9
2	3	8	11
3	3	10	13
4	3	10	13
5	3	10	13
6	3	10	13
7	3	10	13
8	3	10	13
9	3	10	13
10	3	10	13
11	3	10	13
12	3	10	13
13	3	10	13
14	3	10	13
15	3	10	13
16	3	10	13
17	1	10	11
18		10	10
19		10	10
20		10	10
21		10	10
22		1	1
TOTALS	55	205	260

E.29

an estimate based on our present appreciation of the geometry of the orebody and the most probable results of a detailed study.

This schedule calls for two years of pre-revenue stripping at a three million ton-per-year (tpy) rate. This is followed by two years of increasing ore deliveries (plus stripping), after which the total movement of ore and waste would stabilize at 13×10^6 tpy, consisting of 10×10^6 (tpy) of ore and 3×10^6 tpy of waste.

This is assumed to be done on a three shifts per day, five days per week basis, with an allowance of 10 days per year for holidays and 10 days per year lost due to weather and other work stoppages. The schedule, therefore, calls for a 240-day (at three shifts per day) working year for mine operation or 720 shifts per year.

When production has reached the 13 million tpy level, an average daily movement of 54,167 tons will be required. Of this, 41,667 tpd will be ore and 12,500 tpd will be waste on average. Mine equipment and shops will be sized around these parameters.

4.5.2.2 Mill Operations

The mill will be designed to operate with three shifts around the clock all year long. Practically, however, an estimated 35 days will be lost to holidays, other work stoppages, and maintenance. In this study, a 330-day working year was assumed.

As discussed in section 4.5.2.1 on mine operations, the average daily deliveries to the primary crusher will be 41,667 tpd. On peak days, when all equipment is operating and mining ore, this will be much larger. The primary crusher will have to be manned 240 days a year, and a storage facility for primary crushed ore will have to be provided. This facility should hold about a five-day milling supply, or about 150,000 tons.

Downstream from the storage facility at the secondary crushing and ore-grinding stations, the mill would be designed to handle 10 million tpy, or 30,303 tpd. These parameters have been used to size the mill.

5.0 COST SUMMARY

The following tabulation (Table 3) summarizes the costs estimated to place this project onstream and to produce a concentrate throughout the operating life of the project. The annual production of concentrate is also indicated.

These costs are developed in more detail in Appendix A.

APPENDIX A

PALMER THORIUM MINE : COST SUMMARY

			-11	-10	-9	-8	-7	-6	-5	-4	-3	-2	-1 (PRE-REVENUE)	1 (REVENUE)	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
CAPITAL COSTS ($ 000 MISSING)																														
EXPLORATION PHASE	(TABLE A-1)		320	1,289	1,152	217																								
PRE-MINE DEVELOPMENT PHASE	(TABLE A-2)					3,128	8,457	8,168	8,906	5,042																				
CONSTRUCTION PHASE	(TABLE A-3)										19,720	55,000	85,000																	
OPERATIONAL PHASE																														
PRE-REVENUE STRIPPING												3,088	3,088																	
REPLACEMENT OF MILL EQUIPMENT														7,000																
ADDITION & REPLACEMENT OF MINE EQUIP.											15,811	7,906				998	494				9,484	4,737				998	494			
TOTAL CAPITAL			320	1,289	1,152	3,345	8,457	8,165	8,906	5,042	35,531	65,994	88,088	7000	7000	7,998	7,494	7000	7000	7000	16,484	11,737	7000	7000	7000	7,998	7,494	7000	7000	7000
DEPRECIATION OF CAPITAL ($ 000 MISSING)																														
PRE-REVENUE EXPENDITURES (20 yr life - ex mine equip.)														10,128																
MILL EQUIPMENT REPLACEMENT														7,000																
MINE EQUIPMENT														2,047																
TOTAL DEPRECIATION														19,175																
OPERATING COSTS ($ 000 MISSING)																														
MILL (TABLE A-4)																														
LABOR														6,424																
M & S														12,848																
MINE (TABLE A-5)																														
LABOR												1,335	1,355	4,004	6,006															
M & S												1,753	1,753	5,180	7,770															
TOTAL OPERATING												3,088	3,088	28,456	33,048															
CONCENTRATE PRODUCTION (TONS 000 MISSING)																														
TONS OF CONCENTRATE 1)														42	56	70														
MINE PRODUCTION (TONS 000 MISSING)																														
WASTE													3,000																	1,000
ORE														6,000	8,000	10,000														
TOTAL												3,000	3,000	9,000	11,000	13,000														11,000

NOTES:
1.) (a) Each ton of concentrate requires 142.9 tons of mill feed.
(b) Each ton of concentrate contains 1286 pounds of monazite.
(c) Monazite is 8.2% by weight ThO_2
2.) Pre-revenue stripping - capitalized during this period
3.) Last two years omitted

TABLE 3

PRE-REVENUE ○ REVENUE

YEARS	-11	-10	-9	-8	-7	-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	3) 21	3.) 22	23	24	TOTALS

(-1) 320 1,289 1,152 217 2,978

(-2) 3,128 8,457 8,168 8,906 5,042 33,698

(A-3) 19,720 55,000 85,000 159,720

3,088 3,088 6,176

7,000 — 140,000

15,811 7,906 ... 998 494 ... 9484 4737 ... 998 494 ... (NO REPLACM'T) 40,922

320 1,289 1,152 3,345 8,457 8,165 8,906 5,042 35,531 65,994 88,088 7000 7000 7,998 7,494 7,000 7,000 7,000 16,484 11,737 7,000 7,000 7,000 7,998 7,494 7000 7000 7,000 7,000 7,000 7,000 383,500

(p.) 10,128 ——————————————————— 202,560

7,000 ——————————————————— 140,000

2,047 ——————————————————— 40,940

19,175 ——————————————————— 383,500

6,424 ——————————————————— 128,480

12,848 ——————————————————— 256,960

2)

[1,335 1,355] 4,004 6,006 —————————— 4004 ———— 112,112

[1,753 1,753] 5,180 7,770 —————————— 5180 ———— 145,040

[3,088 3,088] 28,456 33,048 —————————— 28,456 ———— 642,592

(MISSING)

42 56 70 ———————————— 1,358

3,000 ——————————————— 1,000 ———— 55,000

6,000 8,000 10,000 ——————————— 194,000

3,000 3,000 9,000 11,000 13,000 ——————— 11,000 10,000 10,000 10,000 249,000

ton of concentrate requires 142.9 tons of mill feed.
ton of concentrate contains 1286 pounds of monazite.
zite is 8.2 % by weight ThO_2
nue stripping - capitalized during this peri :
o years omitted.

E.33

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ACTIVITY & COST ITEMS :

	YEARS	-11	-10	-9	-8

EXPLORATION PHASE

LAND ACQUISITION

    INITIAL OPTIONS OR LEASE — $539,000 1.)

EXPLORATION GEOLOGY

    INITIAL DRILLING & MILLING TESTS — $470,000 2.) $1,193,000 3.)
    INTERPRETATION — $75,000 4.) $100,000 6.)
    PRE-FEASIBILITY STUDIES — $100,000 5.) $200,000 7.)

OTHER — $300,000 8.)

ANNUAL TOTALS	$319,500	$1,288,690	△ $1,152,143	△ $216,667	△ = DROPOUT POINTS TOTAL $2,977,000

COST ESTIMATE :

1) INITIAL OPTIONS OR LEASE
RIGHT TO DRILL & TEST
RIGHT TO PURCHASE AT LATER DATE AT FAIR MKT.
RIGHT TO ACCESS
    4160 AC. OF WHICH 1280 ARE POTENTIALLY
    MINERAL BEARING ; ALSO TOWN OF PALMER

	3520 AC @ $50/AC	$176,000
PALMER =	640 AC @ $200/AC	128,000
STAFF	3 @ $25,000/YEAR	75,000
+ $20,000/YR EXPENSES/MAN		60,000
LEGAL FEES $100,000 YEAR		100,000
+ CONTRACT REVIEW - CONTRACT		
		$539,000

2) INITIAL DRILLING & MILLING TESTS :
PHASE ONE
15 HOLES CORED TOP TO BOTTOM (700 FT)
WITH GEOP. LOGGING

@ $20/FT CONTRACT	$210,000
ASSAYING WORK SAY 3500 SAMPLES @ $25 EA CONTRACT	88,000
3 GEOLOGISTS @ $25,000/YR. FOR 9/10 OF A YEAR	68,000
PLUS $20,000/YR/MAN EXPENSES FOR 9/10 OF A YEAR	54,000
PRELIMINARY MILLING TESTS	50,000
	$470,000

3) INITIAL DRILLING & MILLING TESTS
PHASE TWO
85 HOLES : 1) 22 CORED TOP TO
BOTTOM (700 FT) WITH GEOP.
LOGGING - 7700 ASSAYS @ $20 FT

	$308,000
$25/ASSAY	193,000
2) 63 HOLES - GEOP. LOGS ONLY 700 FT @ $7.00/FT.	309,000
MILL TESTS SAY	100,000
3 GEOLOGISTS @. SEE ABOVE	157,000
PLUS EXPENSES	126,000
	$1,193,000

4) INTERPRETATION OF PHASE ONE
DRILLING
CONSULTANT FEE BASED ON RECENT EXPERIENCE $75,000

5) PRE-FEASIBILITY STUDIES PHASE ONE
CONSULTANT FEE - RECENT EXPERIENCE   100,000

6) INTERPRETATION OF PHASE TWO
DRILLING
CONSULTANT FEE - RECENT EXPERIENCE   100,000

7) PRE-FEASIBILITY STUDIES PHASE TWO
CONSULTANT FEE - RECENT EXPERIENCE   200,000

8) OTHER
EXPLORATION MANAGEMENT, G & A, ETC.
    $100,000/YR.   300,000

TOTAL $2,977,000

△ DROPOUT POINTS

E.A.1

ACTIVITY & COST ITEMS:	YEARS	-8	-7	-6	-5	-4

PRE-MINE DEVELOPMENT PHASE

LAND AQUISITION
    LAND PURCHASE OR LEASE — $9,480,000 ——— 1) ——— $11,750,000 ——— 2)

MINE GEOLOGY
    OREBODY DELINEATION & GRADE
    DISTRIBUTION DETERMINATION — $2,276,000 — 3)
    BULK SAMPLING FOR MILL DESIGN — $4,400,000 — 4)

TECH. & ECONOMIC FEASIBILITY STUDIES
    MINE DESIGN & MINING COSTS — $450,000 — 5)
    MILL DESIGN & MILLING COSTS — $2,930,000 — 6)
    PROJECT COSTS — $1,407,000 — 7)

PERMITS
    ENVIRONMENTAL IMPACT STUDIES — $500,000 — 8)
    SUBMISSION OF PLANS & HEARINGS — $281,000 — 9)
    PERMITS — $225,000 — 10)

ANNUAL TOTALS	$3,128,667	$8,457,333	$8,165,381	$8,905,536	$5,042,083

TOTAL : $33,699,000

△ = DROPOUT POINTS

E.A.3

COST ESTIMATES :

1. LAND PURCHASE OR LEASE - PHASE I
   a) 3520 AC. @ $875/AC $ 3,080,000
      SAY PURCHASED OUTRIGHT
      AT THIS TIME
   b) TOWN OF PALMER -   250 *
      HOMES (640 AC) @ $60,000
      AVERAGE PER HOME = $15,000,000
      1/3 PAID INTO ESCROW ACCOUNT NOW $ 5,000,000
      2/3 PAID IN YEAR - 4
   c) ALSO BUDGET $700,000/YEAR FOR
      RIGHT OF WAYS - LEGAL FEES AND
      MAINTAINENCE OF SMALL LAND-
      MAN STAFF & OTHER LAND PURCHASES
      & RELOCATION EXPENSES $ 1,400,000
                                         $ 9,480,000

2. LAND PURCHASE OR LEASE - PHASE II
   a) FINAL PURCHASE OF PALMER $ 10,000,000
   b) BUDGET $700,000 (See c) above) $ 1,750,000
                                         $ 11,750,000

3. OREBODY DELINEATION & GRADE
   DISTRIBUTION DETERMINATION
   a) 200 DRILL HOLES 700' DEEP
      25% CORED & ASSAYED @ 2' (50 holes)
      75% GEOPHYSICAL LOGS ONLY (150 holes)
      CONTRACT $20/FT. CORE $ 700,000
                $25/FT ASSAY $ 438,000
                $7/FT. PLUG DRILLED $ 735,000
      3 GEOLOGISTS @ $25,000 YR PLUS
                $20,000 EXPENSES $ 203,000
      MILLING INFO FROM CORES
      PHOTOMICROGRAPHS SAY $ 200,000
                                         $ 2,275,000

4. BULK SAMPLING FOR MILL DESIGN
   a) SINK 5' X 10' SHAFT 600' @
      $1,000/FT. - 1/2 YR. CONTRACT $ 600,000
   b) SUPPLY 50 TONS/DAY FOR 1 1/2 YRS.
      LABOR 10 MEN @ $25,000/YR. PLUS
      $20,000/YR. EXPENSES $ 675,000
   c) M & S @ COST OF ABOVE LABOR $ 675,000
   d) PURCHASE & OPERATION OF PILOT MILL
      50 TPD @ $20,000/DAILY TON $ 1,000,000
   e) OPERATE 10 MEN (See above) $ 675,000
   f) M & S $ 675,000
   g) OPERATING PERMITS SAY $ 100,000
                                         $ 4,400,000

5. MINE DESIGN & MINING COSTS
   a) INTERPRET OREBODY, MINE DESIGN, COST
      OPERATION. REQS. 3 GEOL/ENGR.,
      2 DRAFTSMEN, 1 SEC. EQUIVALENT
      CONTRACT PRESENTLY BEING DONE BY
      GOLDER ASSOC. EXPENSES & FEES $ 450,000

6. MILL DESIGN & MILLING COSTS
   a) FLOW SHEET, EQUIPMENT SELECTION
      COSTS (to feasibility standards) TO BE
      DONE BY ENGR. CONST. FIRM. ESTIMATE
      COST 25 MAN YEARS @ 2.5 X PAYROLL
      COST: 25 x $30,000 x 1.25 x 2.5 $ 2,344,000
      EXPENSES @ 25% OF LABOR $ 586,000
                                         $ 2,930,000

7. PROJECT DESIGN & COSTS
   a) INFRASTRUCTURE DESIGN & COSTS,
      ECONOMIC ANALYSIS, ETC.
      SAY 10 MAN YEARS @ 2.5 PAYROLL COSTS
      10 x $30,000 x 1.25 x 2.5 $ 938,000
      EXPENSES @ 50% OF LABOR $ 469,000
                                         $ 1,407,000

8. ENVIRONMENTAL IMPACT STUDIES
   a) COLLECTION OF BASE DATA & REPORT
      BASED ON MINING & MILLING PLANS,
      RECENT EQUIVALENT STUDY.
      COST APPROXIMATELY $ 500,000

9. SUBMISSION OF PLANS & HEARINGS
   a) WILL REQUIRE A STAFF OF 5 MEN
      1.25 YEARS OR 6.25 MAN YEARS @
      $45,000 MAN YEAR PAYROLL &
      OTHER EXPENSES $ 281,000

10. OBTAINING PERMITS
    a) SAME STAFF LEVEL AS 9.)(above)
       FOR 1.0 YEAR $ 225,000

                           TOTAL $ 33,699,000

* According to Marquette County records (July 1978) there are
  222 buildings in Palmer of which 213 are residences. For
  this study we have assumed 250 buildings.

E.A.5

ACTIVITY & COST ITEMS:	YEARS	-3	-2	PRE-REVENUE O REVENUE -1	
CONSTRUCTION PHASE					
MILL CONSTRUCTION		$4,520,000	$5,000,000	$5,000,000	1.)
FINAL DESIGN, PROCUREMENT & CONSTRUCTION MANAGEMENT					
CONSTRUCTION OF MILL AND OTHER FACILITIES		$10,000,000	$50,000,000	$80,000,000	2.)
CONSTRUCTION OF MINE FACILITIES		$5,200,000			3.)
SHOP & OFFICE & PIT POWER					
DISTRIBUTION SYSTEM					
ACCESS ROADS - PIT					
ANNUAL TOTALS		$19,720,000	$55,000,000	$85,000,000	TOTAL $159,720,000

1.) COMPLETE ENGR SERVICES DURING CONSTRUCTION PHASE. FINAL DESIGN-PROCUREMENT AND CONSTRUCTION MANAGEMENT AT 10% OF INSTALLED COST OF ITEMS BEING CONSTRUCTED
$145.2 \times 10^6 \times 0.10$ $14,520,000

2.) CONSTRUCTION OF MILL
  a.) PRIMARY CRUSHER
    @ $296/DAILY TON (REF. SUNNYSIDE STUDY)
    × 41667   12,350,000

  b.) STORAGE 150,000 WITH RECLAIM SYSTEM (REF. PFB)
    STACKING BELT - 1,000,000
    RECLAIM HOPPERS- 500,000
    OTHER (SITE, CONTROLS, ETC) 400,000
          1,900,000   1,900,000

  c.) SECONDARY CRUSHERS & GRINDING SYSTEM @ $1400/DAILY TON (REF. NACAZARI) × 30303   42,424,000

  d.) GRAVITY & MAGNETIC SYSTEM USE $2000/DAILY TON (REF. PFB & SUNNYSIDE) 2000 × 30300   60,606,000

  e.) TAILINGS DISPOSAL $100/DAILY TON (REF. PFB, SUNNYSIDE)   3,030,000

$120,310,000 \times 1.15$ INFLATION 1977 TO 1978 USE 140,000,000 = 4620/DAILY TON (30,000 TPD)

3.) CONSTRUCTION OF MINE FACILITIES
  SHOP × OFFICE 100' × 300'
  $70/SQ.FT. =   2,100,000
  PLUS SHOP & WAREHOUSE & OFFICE EQUIPMENT SAY (REF. PFB. SUNNYSIDE)   1,000,000
  PIT POWER   3,100,000
  @ 400,000 (REF. PFB SUNNYSIDE)   400,000
  ROADS AT SAY (REF. PFB S'SIDE) 1,000,000   1,000,000
        4,500,000 × 1.15 (1977 TO 1978) $5,200,000

TOTAL $159,720,000

E.A.7

OPERATIONAL PHASE: ESTIMATED ANNUAL COSTS - MILL OPERATIONS
(30,000 TPD - 10,000,000 TPY)

FUNCTION	LABOR Per Shift	Total x 4	ANNUAL		
			Labor Cost	Materials & Supplies	Total Direct Oper. Cost
PRIMARY CRUSHER	2	8			
CRUSHED ORE STORAGE & RECLAIM SYSTEM	4	16			
SECONDARY CRUSHING & FINE ORE STORAGE	3	12			
GRINDING	4	16			
GRAVITY SEPARATION CIRCUIT	4	16			
MAGNETIC SEPARATION CIRCUIT	4	16			
FILTRATION CIRCUIT & MILLWATER RECLAMATION & MAKEUP	2	8			
CONCENTRATE STORAGE & LOADOUT	2	8			
TAILING DISPOSAL SYST.	8	32			
SUBTOTAL OP. LAB.	33	132			
MILL MAINTENANCE (INCLUDING WAREHOUSE & MACHINE SHOP, etc.)	30	120			
GENERAL & ADMINISTRATIVE SHIFT SUPER.	6	24			
LAB. (SINGLE SHIFT)	6	6			
OTHER (MIN. SUPER, MIN. ENGR., PUR. AGENT CLERKS, ETC.) (SINGLE SHIFT)	10	10	(x $22,000 PY/MAN)	(x 2 LABOR*)	
		292	6,424,000	12,848,000	19,272,000

*LABOR = 1/3 TOTAL DIRECT COSTS
 @ AVG. 73/SHIFT

ANNUAL CAPITAL ADDITIONS &
REPLACEMENT COSTS @
5% INITIAL COST OF MILL
(20 YEAR AVG. LIFE.)
.05 x $140,000,000 = $7,000,000 PY E.A.9

TOTAL DIRECT MILLING
COSTS $19,272,000

OR 1.93/TON
MILL
FEED

OPERATIONAL PHASE: ESTIMATED ANNUAL CAPITAL AND OPERATING COSTS - MINE OPERATIONS (54,167 TPD - 13,000,000 TPY ORE & WASTE)   TABLE A5

FUNCTION	TYPE	No. REQD.	COST/UNIT READY TO WORK	TOTAL COST	LIFE YEARS	ANNUAL DEPRECIATION	ANNUAL MATERIAL & SUPPLIES	NO MEN/ SHIFT	TOTAL	ANNUAL LABOR COST
MINING ORE & WASTE DRILL & SHOOT	9" DOWNHOLE HAMMER	3	300	900	20	40	968	3	9	
	SLURRY TRUCK	2	70	210	10	21	600 (EXPLOS) 40	4	12	
	FLAT RACK	1	13	13	10	1.3	5	1	3	
LOADING	PH-2300 15 YD. SHOVEL	3	2,800	8400	20	420	1,175	6	18	
	DART-600 15 YD. FRONT END LOADER	3	400	1200	5	240	553	3	9	
HAULING	WABC 120 T ELECTRIC WHEEL TRUCK	18 (6/SHOVEL)	530	9540	10	954	2,820	18	54	
BULL DOZING	CAT D9 TRACK DOZER	3	300	900	10	90	415	3	9	
	MICH 380 RUBBER TIRED DOZER	3	330	990	10	99	484	3	9	
ROAD MAINTENANCE	CAT 16 G	2	180	360	10	36	101	2	6	
	10,000 GAL. WATER TRUCKS	2	100	200	10	20	65	2	6	
	2 YD. HYD. BACKHOE	1	100	100	10	10	46	1	3	
MINE DEWATERING	ELEC. CENTRIF. PUMPS - 6"	3	10	30	5	6	70			
	PORTABLE 6" DIESEL PUMPS	3	20	60	5	12	20			
	WATERLINE 6" & COUPLES	2000'	—	30	10	3	10	2	6	
EQUIPMENT SERVICE	FUEL TRUCK	2	85	170	10	17	10	2	6	
	LUB TRUCK	2	33	66	10	6.6	12	4	12	
OPERATIONS LABOR POOL	PICKUPS CARRYALL	2 3	6 9	12 27	5 5	2.4 5.4	4	6	18	
OPERATION SUPERVISION	PICKUPS	6	6	36	5	7.2	5	3	9	
MINE EQUIPMENT SHOP NOTE: BASIC SHOP EQUIPMENT PURCHASED DURING CONSTRUCTION	70 T CRANE	1	200	200	20	10	20	23 (45 PIECES OF MAJOR EQUIP. X 0.5 MEN/)	69	
	TIRE TRUCK	1	50	50	10	5	7			
	OTHER TRUCKS I.E. WELDING ETC.	—	100	100	10	10	14 300 (SHOP O'HEAD)			
G & A	SEDANS PICKUPS	8 6	7 6	56 36	5 5	11.2 7.2	16 5	10	10	
OTHER	RADIOS	—	—	40	5	8	5	5	5	
TOTAL $				$ 23,717 [4]		$ 2047.3	$ 7770 [3]		273	$ 6,006 [2]

1. $ 22,000 AVG. PAYROLL COST/MAN YEAR

2. $ 0.46/T

3. $ 0.61/T

4. ~ $2.00/ANNUAL TON

APPENDIX B

Daily Mine Production Requirements

The annual daily and shift capacity of the loader will form the basis for the mine production schedule. The mine will work five days per week and three shifts per day (allowing weekends for additional maintanence).

The following number of working days will be assumed:

(52 x 5) - 10 holidays - 10 days lost to weather
and other stoppages;

or 240 days

or 240 x 3 = 720 shifts

The maximum average daily requirement in production year three is therefore: $\frac{13,000,000}{240}$ = 54,167,880 ore and waste.

Ore is $\frac{10}{13}$ x 54,167 = 41,667 TPD

Waste is $\frac{3}{13}$ x 54,167 = 12,500 TPD

Summary:

	Per Day	Per Shift
Ore and Waste	54,167	18,056
Waste	12,500	4,168
Ore	41,667	13,888

Note: Occasionally all the loaders will be loading ore only (as at time the opposite will be true). Therefore, the primary crushing facilities at the mill should be sized to this case.

Equipment Selection

Loading Ore and Waste

Consider 15 cubic yard shovels. These shovels will load between 11,000 and 14,000 tons per operating shift. Considering material types and weather a 12,000 ton per shift figure seems reasonable. Mechanical and operating delays will reduce this output to about 60 percent on an average. The average output will then be 12,000 x .6 = 7,200 TP Shift or:

$$\frac{7,200}{12} = 480 \text{ TP shift/yard of rated bucket capacity}$$

which agrees quite well with industry experience. To produce 18,056 TP shift, $\frac{18.056}{6 \times 12,000} = 2.51$ or 3 shovels will be required when the mine is at maximum in revenue year 3.

Examining Table 1 we note:

Year	Ore & Waste Tonsx10^6	Shifts No/Day	Shifts Total/Year	Tons/Shift Reqd.	Reqd. No. & Calc.	Shovel Sched.
-2	3	1	240	12,500	1.7	2
-1	3	1	240	12,500	1.7	2
1	9	2	480	18,750	2.6	3
2	11	3	720	15,277	2.1	3
3	13	3	720	18,056	2.5	3
17	11	3	720	15,277	2.1	3
18	10	3	720	13,889	1.9	2
22	1	1	240	4,166	0.8	1

AIME Mining Handbook Pool

Private reports: Comparison Western U.S. Copper Mines, 1976

E.B.2

Hauling Ore and Waste

At this point, assume six 120 ton trucks per shovel including an allowance for spares. This is based on the following analysis:

		MIN	CUMMIN
Spot		0.50	
Load 6 swings @ 0.50/swing		3.00	3.50
Haul 1.6 M. @ 10 MPH		9.60	
Dump		1.00	
Return 1.6 M. @ 14 MPH		6.90	21.00
Cycle Time		21.00	

Trucks/Shovel $\dfrac{21.00-3.50}{3.50} = 5$ +1 spare truck

Therefore, the basic loading and hauling module will be one 15 cubic yard shovel (P&H-2300) and six 120 ton trucks (four or five operating 1 and 2 spares). (Wabco 120 electric wheel trucks will be considered.

Drilling Ore and Waste for Blasting

In this study the use of one drill for each shovel has been assumed.

APPENDIX C

Summary of Major Assumptions Used in this Study

1. That the Upper Godrich Quartzite:

    a. is shaped approximately as indicated by Gair (1975, USGS Prof. Paper 769)
    b. has a more or less evenly distributed grade of 14.7 pounds of monazite per ton
    c. the monazite is 8.2 percent by weight ThO_2

2. That the monazite can be concentrated by milling and that 61 percent by weight can be recovered in the concentrate.

3. That land and mineral rights costs will be approximately as indicated on Table A-2-1.

4. That mill construction costs will be approximately as indicated on Table A-3.

5. That the materials and suppliers component of the direct milling costs will be approximately twice the labor component.

APPENDIX F

ELASTICITY OF BYPRODUCTS FROM THORIUM INDUSTRY

ELASTICITY OF BYPRODUCTS FROM THORIUM INDUSTRY

INTRODUCTION

In the event of large scale thorium production, rare earths, titanium, zirconium, hafnium and columbium will be produced as byproducts. The purpose of this report is to analyze the effect of these minerals on their respective markets. The report will define all byproducts, discuss current sources and sources from the thorium industry. Current and prospective markets will then be analyzed to determine the effect of the thorium industry on byproduct credit.

BACKGROUND

Rare Earths

The term "rare earths" came into existence in 1794 when Johann Gadolin assigned it to some poorly understood substances. The term "earth" was then applied to what is now classed as an element. In modern times, rare earths are considered to be the metal oxide form of the element. However, commercially the phrase "rare earth" is used as an adjective (i.e., rare earth oxide, rare earth metal, rare earth chloride, etc.). To avoid confusion, the terms "lanthanide" and "Lanthanoms" have been proposed by V. M. Goldschmidt and J. K. March, respectively, to mean the element form of the rare earths. The term "lanthanides" appears to be gaining favor in scientific circles, but commercially the generic term "rare earths" will probably persist for a long time.[1]

The lanthanides comprise the 15 elements having atomic numbers 57 to 71, and include, in order, lanthanum (La), cerium (Ce), proseodymium (Pr), neodymium (Nd), promethium (Pm), samarium (Sm), europium (Eu), gadolinium (Gd), terbium (Tb), dysprosium (Dy), holmium (Ho), erbium (Er), thulium (Tm), Ytterbium (Yb), and lutetium (Lu). Yttrium is also classified as a rare earth element because of its properties despite the fact its atomic number is 39.

The lanthanide series is divided into two groups. The first seven members (La through Eu) are referred to as the cerium subgroup. Similarly, the remaining eight elements (Gd through Lu) together with yttrium are

called the yttrium subgroup. The two subgroups are also referred to, respectively, as the "light" and "heavy" rare earth elements.[1]

Eight of the rare earths, plus a natural mixture, are commercially important. These include cerium, europium, lanthanum, samarium, yttrium, gasolinium, praseodymium, and neodymium.[2]

The rare earths are found in many minerals and are actually more common than several more familiar elements. Monazite sand is a very important source of rare earths. Placer deposits that contain monazite usually contain ilmenite, rutile, and zircon also.[3]

Titanium

Titanium is also one of the earth's more abundant elements. The most important ores of titanium are ilmenite ($FeTiO_3$) and rutile (TiO_2). Although ilmenite and rutile commonly occur together, rutile is much less abundant than ilmenite.[3] Ilmenite may vary widely in grade, but its theoretical TiO_2 content is 52.8%. Rutile must average 57.4% TiO_2.

Zirconium and Hafnium

Zirconium is a transition metal with properties very similar to hafnium also a transition metal. Chief ores of zirconium are zircon ($ZrSiO_4$) and baddeleyite (ZrO_2). Hafnium is present in almost all zirconium ores in a ratio of about 1:50.[3] Currently marketed zircon contains about 66% ZrO_2 and HFO_2.

Columbium--Background

Columbium is a silver-gray, lustrous metal with an atomic number of 41 and an atomic weight of 92.91. The principal minerals of columbium are columbile and pyrochlore. Pyrochlore concentrates contain 50-55% Cb_2O_5 while columbile concentrate is sold on the basis of 65% Cb_2O_5.

Uranium

Uranium is the first radioactive element to be discovered. It is found in the mineral pitchblend (U_3O_8), and carnotite ($K_2(UO_2)(VO_4)_2 \cdot 8H_2O$). Large scale commercial interest in uranium dates from World War II and the development of the atomic bomb.

Natural uranium contains the isotopes U_{238} (99.285 percent), U_{235} (.71 percent), and U_{234} (.005 percent). U_{235} is the only natural occurring fissionable isotope. U_{238} is of major significance because it yields plutonium.

CURRENT SOURCES

Rare Earths

Current mineral sources of rare earths are bastnasite, monazite and xenotime. Monazite and bastnasite contain only negligible amounts of the "heavy" lanthanums, but consist primarily of cerium subgroup elements. Xenotime, however, contains the yttrium subgroup minerals.[1]

Bastnasite rare earth oxide concentrates are produced by open pit mining from a deposit in Mountain Pass, California, by Molycorp, Inc., a subsidiary of Union Oil of California. Mountain Pass also supplies purified rare earths including cerium, lanthanum, neodymium/praseodymium, gadolinium/samarium, and europium oxide. In 1977, 14,900 tons of contained rare earth oxides were produced by Molycorp with exports consuming 40%.[2]

Monazite sand is also an important commercial source of rare earths. Australia and Malaysia are the main sources of monazite, with U.S. imports totaling 3,300 short tons of rare earth oxides (REO). Monazite is also recovered domestically from placers near Jacksonville, Florida, by Humphrey's Mining Company. About 10% of the U.S. market for monazite is supplied by domestic resources. Davison Chemical Division, W. R. Grace & Co. at Chattanooga, Tennessee, is the only monazite processor in the U.S.[2]

Xenotime concentrates are processed primarily for yttrium oxide content. Canada and Malaysia are the principal suppliers. Only a small amount of the total domestic REO tonnage is shipped from these sources.[2]

Titanium

Major sources of titanium are placer sands which contain varying amounts of ilmenite and rutile. About 70% of the ilmenite produced in the U.S. comes

from three mines, two in Florida and one in New York. Other mines in New Jersey and Florida produce the remaining 30%. Ilmenite output from the rest of the world comes mostly from Canada, Norway, Australia, and the USSR.[4] In 1977, total domestic mine production from ilmenite was 540,000 tons. Imports of ilmenite amounted to 256,000 tons. Total U.S. consumption of titanium in 1977 was 54,000 tons.[5]

Zirconium

Zircon is the only zirconium ore of commercial significance. It is recovered as a byproduct of titanium mining. Zircon is recovered from mineral sands by Humphrey's Mining Company for Du Pont near Boulougne, Florida; firms in Australia, Japan, and Egypt also process sand for its zircon content.[6] In 1977, total domestic demand for zirconium was 4,300 tons.[7]

Columbium

The United States relies on imports for its supply of columbium-bearing ores and concentrates. Brazil, Canada, Thailand, and Nigeria accounted for the majority of columbium imports in 1977. Total imports in 1977 amounted to 2,477,000 pounds contained columbium.[11]

Uranium

About 98% of domestic reserves at $10/lb U_3O_8 are in sandstone-type deposits mainly in the Ambrosia Lake district of New Mexico and in Wyoming. Uranium is also recovered as a byproduct of copper and phosphate fertilizer production.

Most of the large economic deposits have a uranium content of .1% to .3% of U_3O_8 averaging .2%. "Yellowcake", the term applied to the uranium concentrate produced at the mills, is chemically in the form of ammonium diuranite or sodium diuranite. Most concentrates contain a minimum of .75% U_3O_8. The cost for yellowcake in December 1977 was $43.20/lb of U_3O_8.

POTENTIAL SOURCES FROM ThO_2 PRODUCTION

Thorium deposits may also contain significant deposits of rare earths, titanium, zirconium and columbium. During mining and processing, rare earths will be liberated and are potentially recoverable as byproducts. Deposits with this potential include Lemhi Pass, Palmer Michigan, Bald Mountain, Mountain Pass, and all of the placer deposits.

Rare Earths

Thorite veins such as those found in Lemhi Pass, Idaho, Wet Mountain, Colorado, and Powerhorn, Colorado are potential sources of thorium and rare earths. The rare earths are contained largely in the thorite, although small amounts of monazite and zenotime are present. Assuming the rare earth content is equal to the thorium content, approximately 70,000 tons of REO's could be mined with the thorium. An estimate based on spectographic analysis showed yttrium was the dominant element followed first by yttrium subgroup metals and then the light subgroup metals.

Piedmont placer deposits of North and South Carolina constitute another potential source of rare earths. Monazite is by far the most common mineral. There are an estimated 123,000 tons of rare earth oxides available from these placer sources along with about 11,000 tons of ThO_2. A typical breakdown of the rare earth composition of monazite is shown in Table F.1.

TABLE F.1. REO Content in Monazite[8]

Rare Earths	(%)
Lanthanum	23
Cerium	45
Proseodymium ⎫ ⎬ Dysprosium Neodymium ⎭	25
Samarium	3
TOTAL Cerium Group	96
Ytterbium	2
Other	2
TOTAL Yttrium Group	4

The Mountain Pass, California deposit is currently mined for rare earth elements with a potential for at least 5 million tons. The deposit also contains some thorium estimated at about 28,500 tons. The rare earths portion consists of 56% cerium, 28.6% lanthanum, 4 1% praseodymium, 10.5% neodymium, .6% samarium, and .2% gadolinium.[3] About 2 million tons could be processed as a byproduct of thorium production.

Monazite also exists in conglomerate deposits of Palmer, Michigan, and Bald Mountain, Wyoming. The Palmer deposit contains over 1 million tons of rare earths. Bald Mountain contains an estimated 16,000 tons of rare earth oxides.

The thorium-bearing carbonatite of Iron Hill, Colorado contains rare earths also. Rare earths are estimated at .8% of the deposit and constitute approximately 810,000 tons.

Titanium

Large-scale production of thorium may, at some period of time, make mining of domestic placer deposits desirable. Beach and stream placers commonly contain both rutile and ilmenite. Mining of placers could result in 400,000 tons of ilmenite (tons TiO_2). Rutile is much more versatile than ilmenite, but far less common. Mining of placers could result in the production of approximately 23,000 tons of rutile (tons TiO_2).[4]

Zirconium and Hafnium

Approximately 45,000 tons of zircon are contained in domestic placers. Zircon contains about 66% ZrO_2 including HfO_2.[6] Zirconium and hafnium could be recovered as a byproduct of mining placer sands.

Columbium

The carbonatite deposit of Powerhorn, Colorado (Iron Hill) is a potential source of byproduct columbium.

The major columbium-bearing mineral is pyrochlore; however, the thorium-bearing minerals are not clearly identified. In all likelihood, pyrochlore also contains some thorium. Unfortunately, other thorium-bearing minerals may be present in the carbonatite. Thus, recovery of both thorium and columbium in a single concentrate may be difficult and, if accomplished, it will result in a lower grade columbium-bearing concentrate than pure pyrochlore. Current chemical processing techniques do not appear to be able to economically recover columbium from such a low grade starting material. However, future research regarding the deposit, benefication techniques, and refining methods might alter the situation. Thus, Iron Hill is still a potential source of columbium as a byproduct of thorium production. As much as 10,000 tons per year of columbium pentoxides could be liberated as a byproduct of thorium production.

Placer deposits of Bear Valley, Idaho are also a potential source of columbium. The mineral occurs in the form of columbite. In the event of large-scale thorium production, a possible 330 tons per year of columbium pentoxide could be liberated as a byproduct.

Uranium

The primary thorium deposit associated with uranium is the Conway Granite reserve located in New Hampshire. Output is approximately 800,000 lb/yr of contained U_3O_8. The product will be marketed as yellowcake and sold to uranium production facilities. Approximately 8,750,000 tons of uranium are contained in this deposit.

Uranium is also contained in monazite from conglomerate and placer deposits. There is approximately 23 parts thorium to 1 part uranium, or 6,000 tons of U_3O_8. It is currently not economical to recover placer deposits. Due to the low grade nature of conglomerate deposits, uranium is not economical to recover.

CURRENT AND PROSPECTIVE MARKETS

Rare Earths

Many of the uses of rare earth products are listed in Table F.2. Industrial applications of the rare earths have increased markedly in recent years. The traditional uses of rare earths, including ceramic additives and polishing compounds are, of course, still important, but new industry consumes the bulk of production and consumes much of the rare earth market.

Table F.2. Uses of Rare Earth Elements and Yttrium

Metallurgy	Alloying agents in iron and steel, superalloys, and pyrophoric alloys, lighter flints, pure metals for research.
Glass	Polishing, decolorizing, coloring, filters, optical and photochromic glass (camera lenses).
Ceramics	Colorants for enamels and glazes, coatings, refractories and stabilizers.
Illumination	Carbon arcs, lasers, fluorescent and mercury vapor lamps, phosphors (x-ray intensifiers, display, and color television).
Electronic	Capacitors, cathodes, electrodes, semiconductors, thermistors, magnets, computer components (garnets and ferrites) and memories.
Nuclear	Control rods, burnable poisons, dilutants, shielding, radioactive heat and power sources, detectors and counters.
Chemical	Catalysts, pharmaceutical, water treatment, chemical processing and analysis, shift reagents and tags in organic and biological chemistry.
Other	Jewelry, photography, lubrication, thermometers, paint and ink dryers, textiles.

Mineral Facts and Problems, 1975 Ed.

Expansion of the rare earths industry depends on the development of new uses. Although demand for rare earths and yttrium grew rapidly during 1965-1974, the outlook for future growth of the U.S.'s rare earths industry is for a more moderate rate of expansion. Projected demand for rare earth oxides in

the year 2000 is between 24,000 and 50,000 short tons with 34,000 short tons most probable.[2] The development is hampered in part by the lack of fundamental information on the properties of rare earth compounds and metal. Only a few years ago most of the high purity individual rare earth elements were little more than laboratory curiosity. Other factors affecting the expansions of rare earths usage are consumer hesitancy to accept them, loss of established markets resulting from limited growth of petroleum refining, and substitution of rare earths and yttrium by cheaper or superior materials.[10]

Attainment of the high rare earth forecast depends primarily on continued expansion of established markets, mainly iron and steel industry. Recent potential markets include: 1) new catalysts for automobile exhaust systems and for chemical processing purposes, 2) permanent magnets for computer and electronic systems for trucks and automobiles, and 3) hydrogen storage and fueling systems for equipment used in confined areas.[9]

Titanium

Major markets for titanium are in the manufacture of titanium dioxide pigments which accounts for 85% of the industrial value. Other uses include titanium sponge, welding rod coatings and carbides, and ceramic and glass formulations.

Titanium dioxide pigment and other titanium compounds have many uses including rubber tires, floor coverings, printing ink, porcelain enamels, wall covering, oil cloth and other coated fabrics, and roof coatings. Several thousand tons of titanium dioxide are used annually in the production of welding rod coatings, glass fibers, ceramic capacitors, and electromechanical transducers. Commercial carbide cutting tools contain 8% to 85% titanium carbide. Organotitanium compounds are used as catalysts for various polymerization processes, as water repellants, and in dyeing processes.

Titanium metal is primarily used in aerospace applications including aircraft and guided missile assemblies, spacecraft, and turbine engines for aircraft. The chemical and electrochemical processing industries also use some metal.

Present technology requires ilmenite or titanium slag for use in making titania pigment via a sulfate process. A chloride process is used to make pigment from rutile. The chloride process is environmentally cleaner, but problems arise since rutile is of limited availability and higher cost. Rutile substitutes, called synthetic rutile, however, can be made from ilmenite. Semi-commercial plants are in India, Japan, and Australia. Research is also being cnducted in the U.S. to determine an economic method of producing domestic rutile substitutes.[4]

The demand for titanium in the U.S. in the year 2000 is projected to range between 1.1 million and 2.0 million tons. The figure of 1.45 million tons is projected as most probable. This is over a 90% increase in demand from 1977. This increase is due to large demands for metal from turbine aircraft engines. Growth in the use of titanium pigments is due largely to increased population and industrial demand.[5]

Zirconium and Hafnium

The consumption zirconium has been chiefly in the form of the mineral zircon and as zirconium oxide. Zircon is recovered primarily as a byproduct of titanium mining.

The major use of zircon is for facings on foundry molds. Here, cost controls competition from substitutes. Refractories also employ zircon for specialized application. Small but important uses of zirconium compounds are in chemical products. Another minor use for zircon is in the production of metal and alloys used as structural material for nuclear reactors and chemical processing equipment.

Hafnium is not removed from zirconium for most uses because its presence is not detrimental. However, the two elements must be separated for nuclear purposes. Hafnium is primarily used in the form of the metal. Most metal is used in control rods in nuclear reactors, but minor quantities are used in flashbulbs, optical glass, and in refractory alloys. Small amounts of hafnium oxide are used in research.[6]

U.S. demand for zirconium is currently about 4,250[7] tons. By the year 2000, U.S. consumption is estimated at 130,500 tons and 312,000 tons with 223,000 tons estimated as most probable.[6]

Columbium

Columbium products for trade and industry consist mainly of pyrochlore and columbite concentrates for use by processing industries. Columbium is also used in the form of ferrocolumbium or columbium pentoxide in the manufacture of steel, superalloys, columbium metal, columbium-base alloys, and columbium carbide. The steel industry utilizes columbium to control grain size which, in turn, offers improved strength to weight ratios. Columbium-based steels, alloys, and metals also offer increased corrosion resistance.[12]

U.S. demand for columbium is forecast at 11 million pounds in 1985 and between 15-29.8 million pounds in the year 2000. Forecasts were based primarily on the demand for the different types of steels and the principal end use of each type.[12]

Uranium

Uranium is primarily utilized as nuclear fuel. Despite adverse factors, this continues to be the major use. Other uses include government sponsored programs -- weapons, propulsion, underground tests, research and development, and space applications. Depleted uranium is used as shielding, castings, weapons parts, and ammunition. Uranium also has a number of chemical uses, including catalyst in plastics manufacture, a colorant in glass and ceramics. It is also used as targets for x-ray tubes, electrodes in ultraviolet light sources, and resistors in incandescent lamps.

Nuclear power is expected to assume an increasing share of domestic electric generating capacity. Projections of installed nuclear capacity varies from 380-620 gigawatts electric by the year 2000. Projections for uranium requirements for nuclear uses in the year 2000 vary from 62,000 to 99,000 tons of U_3O_8 annually. The low forecast assumes 4,000,000 MW installed capacity, 2% tails, enrichment plant tails assay, minimal public acceptance, a maximum energy conservation effort, minimal improvement in reactor construction lead times, and uranium shortages. The high forecast assumes 900,000 MWe installed capacity, adequate uranium supply, public acceptance, and decreased lead time for reactor construction.

EFFECT OF ThO$_2$ PRODUCTION

Although our study indicates that byproduct recovery of rare earths is not economically feasible, serious consequences would result if currently subeconomic sources of rare earths were made available. As much as 100,000 tpy of rare earths could be marketed as a result of large-scale thorium production. The market would be flooded unless demand projections were influenced. To accomplish this, lowering the price of the rare earth materials available from thorium production might enable new, cheaper uses for rare earth products to increase demand projections. Most rare earths are marketed as misch metal or as refined rare earth metals and compounds. High prices for processed rare earths are reportedly due to large quantity consumption of chemicals, cost of labor, and substantial plant investment because of relatively low throughput capacity.[10] Thus, it appears that the cost of producing these materials would be only slightly influenced by availability of a cheaper rare earths raw material. Price decreases of rare earth products would not be substantial and demand projections would still be reasonable.

The most likely effect of large-scale production of rare-earths and thorium seems to be the formation of stockpiles resulting in rare earth surpluses. If the price of rare earth raw materials is comparably inexpensive, these stockpiles might be maintained by rare earth processors. However, the stockpiles will probably be maintained by the thorium industry until demand rises sufficiently to utilize supply. With this knowledge, it appears that no byproduct credit is the most realistic assumption for rare earths.

Titanium

The amount of titanium from thorium production could be as high as 23,000 tons of TiO$_2$ equivalent per year. The demand in the year 2000 is estimated at 1.4 million tons of TiO$_2$. Thus, it appears that titanium from the thorium industry could easily be absorbed by the domestic market.

Problems with marketing ilmenite, however, have been realized because of environmental problems encountered during processing. A surplus of tin slag which can be substituted for ilmenite in pigment manufacture has also reduced the value of ilmenite. But the development of a process for converting ilmenite to rutile will tend to increase the value of ilmenite and offset the current downtrend in the value of ilmenite. Thus, the full price of ilmenite shall be credited to the byproducts in this study.

Zirconium

As much as 3,000 tons per year of zirconium would be liberated as a byproduct of domestic placer mining. Since the demand for zirconium by the year 2000 is estimated at 223,000 tons, the cost of zirconium should not be influenced by the relatively small amount of byproduct zirconium from the thorium industry. Full byproduct credit for zirconium can be given.

Columbium

If economical methods of obtaining columbium products from carbonatite are developed, problems with marketing the columbium byproducts could develop. Demand in the year 2000 is forecast at a probable 11,150 tons. This is very similar to the possible 10,000 tons per year production from Iron Hill. Competition from other markets might result in price reductions or stockpiling of some of the resource. However, columbium is a useful alloying agent for steel and its use would expand if the price decreased. In this case, a by-credit of 50% of current market shall be assumed.

Uranium

Production of Conway Granites would release approximately 400 tpy of U_3O_8. The demand in the year 2000 ranges from 30,000 to 99,000 tons of U_3O_8. From these estimates it appears that uranium from Conway Granite should easily be marketed with no effect on price.

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APPENDIX G

FREQUENCY TABLES FOR PRODUCTION COST UNCERTAINTY
ANALYSIS OF DOMESTIC THORIUM DEPOSITS

TABLE G.1. Stockpiles (Chattanooga Sludge)

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
4.30	1	.50	.50	99.50
4.40	0	.00	.50	99.50
4.50	1	.50	1.00	99.00
4.60	2	1.00	2.00	98.00
4.70	4	2.00	4.00	96.00
4.80	2	1.00	5.00	95.00
4.90	5	2.50	7.50	92.50
5.00	3	1.50	9.00	91.00
5.10	6	3.00	12.00	88.00
5.20	6	3.00	15.00	85.00
5.30	8	4.00	19.00	81.00
5.40	11	5.50	24.50	75.50
5.50	12	6.00	30.50	69.50
5.60	9	4.50	35.00	65.00
5.70	14	7.00	42.00	58.00
5.80	10	5.00	47.00	53.00
5.90	14	7.00	54.00	46.00
6.00	11	5.50	59.50	40.50
6.10	6	3.00	62.50	37.50
6.20	4	2.00	64.50	35.50
6.30	6	3.00	67.50	32.50
6.40	12	6.00	73.50	26.50
6.50	8	4.00	77.50	22.50
6.60	5	2.50	80.00	20.00
6.70	5	2.50	82.50	17.50
6.80	7	3.50	86.00	14.00
6.90	7	3.50	89.50	10.50
7.00	4	2.00	91.50	8.50
7.10	4	2.00	93.50	6.50
7.20	1	.50	94.00	6.00
7.30	2	1.00	95.00	5.00
7.40	3	1.50	96.50	3.50
7.50	1	.50	97.00	3.00
7.60	0	.00	97.00	3.00
7.70	2	1.00	98.00	2.00
7.80	0	.00	98.00	2.00
7.90	1	.50	98.50	1.50
8.00	1	.50	99.00	1.00
8.10	0	.00	99.00	1.00
8.20	1	.50	99.50	.50
8.30	0	.00	99.50	.50
8.40	0	.00	99.50	.50
8.50	1	.50	100.00	.00

TABLE G.2. Stockpiles (Curtis Bay--Miamisburg Sludge)

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
2.86	2	1.00	1.00	99.00
2.94	1	.50	1.50	98.50
3.02	2	1.00	2.50	97.50
3.10	2	1.00	3.50	96.50
3.18	1	.50	4.00	96.00
3.26	5	2.50	6.50	93.50
3.34	10	5.00	11.50	88.50
3.42	7	3.50	15.00	85.00
3.50	12	6.00	21.00	79.00
3.58	9	4.50	25.50	74.50
3.66	12	6.00	31.50	68.50
3.74	11	5.50	37.00	63.00
3.82	14	7.00	44.00	56.00
3.90	10	5.00	49.00	51.00
3.98	13	6.50	55.50	44.50
4.06	13	6.50	62.00	38.00
4.14	9	4.50	66.50	33.50
4.22	7	3.50	70.00	30.00
4.30	14	7.00	77.00	23.00
4.38	8	4.00	81.00	19.00
4.46	4	2.00	83.00	17.00
4.54	9	4.50	87.50	12.50
4.62	7	3.50	91.00	9.00
4.70	3	1.50	92.50	7.50
4.78	2	1.00	93.50	6.50
4.86	1	.50	94.00	6.00
4.94	1	.50	94.50	5.50
5.02	0	.00	94.50	5.50
5.10	1	.50	95.00	5.00
5.18	5	2.50	97.50	2.50
5.26	1	.50	98.00	2.00
5.34	0	.00	98.00	2.00
5.42	1	.50	98.50	1.50
5.50	1	.50	99.00	1.00
5.58	2	1.00	100.00	.00

TABLE G.3. Stockpiles (Fernald Sludge)

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
2.02	2	1.00	1.00	99.00
2.08	1	.50	1.50	98.50
2.14	2	1.00	2.50	97.50
2.20	2	1.00	3.50	96.50
2.26	7	3.50	7.00	93.00
2.32	9	4.50	11.50	88.50
2.38	8	4.00	15.50	84.50
2.44	13	6.50	22.00	78.00
2.50	8	4.00	26.00	74.00
2.56	13	6.50	32.50	67.50
2.62	6	3.00	35.50	64.50
2.68	12	6.00	41.50	58.50
2.74	13	6.50	48.00	52.00
2.80	14	7.00	55.00	45.00
2.86	10	5.00	60.00	40.00
2.92	12	6.00	66.00	34.00
2.98	9	4.50	70.50	29.50
3.04	5	2.50	73.00	27.00
3.10	7	3.50	76.50	23.50
3.16	12	6.00	82.50	17.50
3.22	4	2.00	84.50	15.50
3.28	7	3.50	88.00	12.00
3.34	3	1.50	89.50	10.50
3.40	4	2.00	91.50	8.50
3.46	3	1.50	93.00	7.00
3.52	4	2.00	95.00	5.00
3.58	3	1.50	96.50	3.50
3.64	2	1.00	97.50	2.50
3.70	2	1.00	98.50	1.50
3.76	0	.00	98.50	1.50
3.82	0	.00	98.50	1.50
3.88	0	.00	98.50	1.50
3.94	0	.00	98.50	1.50
4.00	1	.50	99.00	1.00
4.06	0	.00	99.00	1.00
4.12	0	.00	99.00	1.00
4.18	1	.50	99.50	.50
4.24	1	.50	100.00	.00

TABLE G.4. Hall Mountain

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
5.60	1	.50	.50	99.50
5.70	1	.50	1.00	99.00
5.80	3	1.50	2.50	97.50
5.90	2	1.00	3.50	96.50
6.00	7	3.50	7.00	93.00
6.10	6	3.00	10.00	90.00
6.20	8	4.00	14.00	86.00
6.30	14	7.00	21.00	79.00
6.40	5	2.50	23.50	78.50
6.50	12	6.00	29.50	70.50
6.60	13	6.50	36.00	64.00
6.70	13	6.50	42.50	57.50
6.80	10	5.00	47.50	52.50
6.90	12	6.00	53.50	46.50
7.00	4	2.00	55.50	44.50
7.10	12	6.00	61.50	38.50
7.20	9	4.50	65.00	34.00
7.30	8	4.00	70.00	31.00
7.40	4	2.00	72.00	28.00
7.50	6	3.00	75.00	25.00
7.60	4	2.00	77.00	23.00
7.70	3	1.50	78.50	21.50
7.80	7	3.50	82.00	18.00
7.90	4	2.00	84.00	16.00
8.00	5	2.50	86.50	13.50
8.10	2	1.00	87.50	12.50
8.20	3	1.50	89.00	11.00
8.30	3	1.50	90.50	8.50
8.40	5	2.50	93.00	7.00
8.50	4	2.00	95.00	5.00
8.60	5	2.00	97.00	3.00
8.70	2	1.00	98.00	2.00
8.80	0	.00	98.00	2.00
8.90	0	.00	98.00	2.00
9.00	0	.00	98.00	2.00
9.10	2	1.00	98.00	1.00
9.20	0	.00	98.00	1.00
9.30	2	1.00	100.00	.00

TABLE G.5. Lemhi Pass (Stage 1)

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
11.10	1	.50	.50	99.50
11.40	0	.00	.50	99.50
11.70	0	.00	.50	99.50
12.00	3	1.50	2.00	98.00
12.30	2	1.00	3.00	97.00
12.60	1	.50	3.50	96.50
12.90	6	3.00	6.50	93.50
13.20	7	3.50	10.00	90.00
13.50	6	3.00	13.00	87.00
13.80	9	4.50	17.50	82.50
14.10	12	6.00	23.50	76.50
14.40	11	5.50	29.00	71.00
14.70	6	3.00	32.00	68.00
15.00	11	5.50	37.50	62.50
15.30	10	5.00	42.50	57.50
15.60	15	7.50	50.00	50.00
15.90	11	5.50	55.50	44.50
16.20	12	6.00	61.50	38.50
16.50	11	5.50	67.00	33.00
16.80	11	5.50	72.50	27.50
17.10	4	2.00	74.50	25.50
17.40	7	3.50	78.00	22.00
17.70	12	6.00	84.00	16.00
18.00	8	4.00	88.00	12.00
18.30	4	2.00	90.00	10.00
18.60	5	2.50	92.50	7.50
18.90	4	2.00	94.50	5.50
19.20	1	.50	95.00	5.00
19.50	2	1.00	96.00	4.00
19.80	1	.50	96.50	3.50
20.10	2	1.00	97.50	2.50
20.40	2	1.00	98.50	1.50
20.70	0	.00	98.50	1.50
21.00	2	1.00	99.50	.50
21.30	0	.00	99.50	.50
21.60	0	.00	99.50	.50
21.90	0	.00	99.50	.50
22.20	0	.00	99.50	.50
22.50	0	.00	99.50	.50
22.80	0	.00	99.50	.50
23.10	1	.50	100.00	.00

TABLE G.6. Lemhi Pass (Stage 2)

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
9.10	3	1.50	1.50	98.50
9.30	2	1.00	2.50	97.50
9.50	3	1.50	4.00	96.00
9.70	3	1.50	5.50	94.50
9.90	0	.00	5.50	94.50
10.10	6	3.00	8.50	91.50
10.30	1	.50	9.00	91.00
10.50	4	2.00	11.00	89.00
10.70	8	4.00	15.00	85.00
10.90	10	5.00	20.00	80.00
11.10	10	5.00	25.00	75.00
11.30	8	4.00	29.00	71.00
11.50	10	5.00	34.00	66.00
11.70	11	5.50	39.50	60.50
11.90	8	4.00	43.50	56.50
12.10	18	9.00	52.50	47.50
12.30	3	1.50	54.00	46.00
12.50	8	4.00	58.00	42.00
12.70	11	5.50	63.50	36.50
12.90	7	3.50	67.00	33.00
13.10	10	5.00	72.00	28.00
13.30	10	5.00	77.00	23.00
13.50	7	3.50	80.50	19.50
13.70	6	3.00	83.50	16.50
13.90	4	2.00	85.50	14.50
14.10	9	4.50	90.00	10.00
14.30	3	1.50	91.50	8.50
14.50	3	1.50	93.00	7.00
14.70	1	.50	93.50	6.50
14.90	2	1.00	94.50	5.50
15.10	2	1.00	95.50	4.50
15.30	0	.00	95.50	4.50
15.50	5	2.50	98.00	2.00
15.70	1	.50	98.50	1.50
15.90	2	1.00	99.50	.50
16.10	0	.00	99.50	.50
16.30	0	.00	99.50	.50
16.50	0	.00	99.50	.50
16.70	0	.00	99.50	.50
16.90	0	.00	99.50	.50
17.10	0	.00	99.50	.50
17.30	0	.00	99.50	.50
17.50	1	.50	100.00	.00

TABLE G.7. Wet Mountain

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
6.10	3	1.50	1.50	98.50
6.30	1	.50	2.00	98.00
6.50	2	1.00	3.00	97.00
6.70	7	3.50	6.50	93.50
6.90	10	5.00	11.50	88.50
7.10	9	4.50	16.00	84.00
7.30	8	4.00	20.00	80.00
7.50	7	3.50	23.50	76.50
7.70	11	5.50	29.00	71.00
7.90	12	6.00	35.00	65.00
8.10	9	4.50	39.50	60.50
8.30	14	7.00	46.50	53.50
8.50	11	5.50	52.00	48.00
8.70	9	4.50	56.50	43.50
8.90	11	5.50	62.00	38.00
9.10	7	3.50	65.50	34.50
9.30	4	2.00	67.50	32.50
9.50	7	3.50	71.00	29.00
9.70	5	2.50	73.50	26.50
9.90	4	2.00	75.50	24.50
10.10	2	1.00	76.50	23.50
10.30	1	.50	77.00	23.00
10.50	7	3.50	80.50	19.50
10.70	4	2.00	82.50	17.50
10.90	1	.50	83.00	17.00
11.10	3	1.50	84.50	15.50
11.30	6	3.00	87.50	12.50
11.50	4	2.00	89.50	10.50
11.70	4	2.00	91.50	8.50
11.90	5	2.50	94.00	6.00
12.10	6	3.00	97.00	3.00
12.30	1	.50	97.50	2.50
12.50	2	1.00	98.50	1.50
12.70	1	.50	99.00	1.00
12.90	0	.00	99.00	1.00
13.10	1	.50	99.50	.50
13.30	0	.00	99.50	.50
13.50	1	.50	100.00	.00

TABLE G.8. Palmer, Michigan

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
9.90	6	3.00	3.00	97.00
10.10	2	1.00	4.00	96.00
10.30	1	.50	4.50	95.50
10.50	1	.50	5.00	95.00
10.70	3	1.50	6.50	93.50
10.90	10	5.00	11.50	88.50
11.10	1	.50	12.00	88.00
11.30	8	4.00	16.00	84.00
11.50	3	1.50	17.50	82.50
11.70	5	2.50	20.00	80.00
11.90	7	3.50	23.50	76.50
12.10	3	1.50	25.00	75.00
12.30	4	2.00	27.00	73.00
12.50	6	3.00	30.00	70.00
12.70	9	4.50	34.50	65.50
12.90	12	6.00	40.50	59.50
13.10	7	3.50	44.00	56.00
13.30	9	4.50	48.50	51.50
13.50	4	2.00	50.50	49.50
13.70	9	4.50	55.00	45.00
13.90	8	4.00	59.00	41.00
14.10	13	6.50	65.50	34.50
14.30	13	6.50	72.00	28.00
14.50	0	.00	72.00	28.00
14.70	9	4.50	76.50	23.50
14.90	9	4.50	81.00	19.00
15.10	4	2.00	83.00	17.00
15.30	8	4.00	87.00	13.00
15.50	0	.00	87.00	13.00
15.70	4	2.00	89.00	11.00
15.90	1	.50	89.50	10.50
16.10	7	3.50	93.00	7.00
16.30	6	3.00	96.00	4.00
16.50	2	1.00	97.00	3.00
16.70	4	2.00	99.00	1.00
16.90	0	.00	99.00	1.00
17.10	2	1.00	100.00	.00

TABLE G.9. Bald Mountain

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
29.00	1	.50	.50	99.50
30.00	0	.00	.50	99.50
31.00	0	.00	.50	99.50
32.00	2	1.00	1.50	98.50
33.00	2	1.00	2.50	97.50
34.00	2	1.00	3.50	96.50
35.00	0	.00	3.50	96.50
36.00	5	2.50	6.00	94.00
37.00	3	1.50	7.50	92.50
38.00	4	2.00	9.50	90.50
39.00	7	3.50	13.00	87.00
40.00	7	3.50	16.50	83.50
41.00	5	2.50	19.00	81.00
42.00	8	4.00	23.00	77.00
43.00	11	5.50	28.50	71.50
44.00	6	3.00	31.50	68.50
45.00	7	3.50	35.00	65.00
46.00	9	4.50	39.50	60.50
47.00	15	7.50	47.00	53.00
48.00	5	2.50	49.50	50.50
49.00	16	8.00	57.50	42.50
50.00	14	7.00	64.50	35.50
51.00	11	5.50	70.00	30.00
52.00	5	2.50	72.50	27.50
53.00	9	4.50	77.00	23.00
54.00	11	5.50	82.50	17.50
55.00	2	1.00	83.50	16.50
56.00	3	1.50	85.00	15.00
57.00	6	3.00	88.00	12.00
58.00	4	2.00	90.00	10.00
59.00	6	3.00	93.00	7.00
60.00	5	2.50	95.50	4.50
61.00	2	1.00	98.50	3.50
62.00	1	.50	97.00	3.00
63.00	3	1.50	98.50	1.50
64.00	0	.00	99.00	1.00
65.00	1	.50	99.50	1.00
66.00	1	.50	99.50	.50
67.00	0	.00	99.50	.50
68.00	1	.50	100.00	.00

TABLE G.10. Bear Lodge--North Block

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
41.00	1	.50	.50	99.50
42.00	0	.00	.50	99.50
43.00	0	.00	.50	99.50
44.00	2	1.00	1.50	98.50
45.00	3	1.50	3.00	97.00
46.00	3	1.50	4.50	95.50
47.00	5	2.50	7.00	93.00
48.00	4	2.00	9.00	91.00
49.00	8	4.00	13.00	87.00
50.00	7	3.50	16.50	83.50
51.00	3	1.50	18.00	82.00
52.00	8	4.00	22.00	78.00
53.00	11	5.50	27.50	72.50
54.00	11	5.50	33.00	67.00
55.00	10	5.00	38.00	62.00
56.00	9	4.50	42.50	57.50
57.00	10	5.00	47.50	52.50
58.00	8	4.00	51.50	48.50
59.00	11	5.50	57.00	43.00
60.00	6	3.00	60.00	40.00
61.00	10	5.00	65.00	35.00
62.00	8	4.00	69.00	31.00
63.00	7	3.50	72.50	27.50
64.00	10	5.00	77.50	22.50
65.00	6	3.00	80.50	19.50
66.00	9	4.50	85.00	15.00
67.00	5	2.50	87.50	12.50
68.00	6	3.00	90.50	9.50
69.00	6	3.00	93.50	6.50
70.00	2	1.00	94.50	5.50
71.00	0	.00	94.50	5.50
72.00	1	.50	95.00	5.00
73.00	4	2.00	97.00	3.00
74.00	4	2.00	99.00	1.00
75.00	1	.50	99.50	.50
76.00	0	.00	99.50	.50
77.00	0	.00	99.50	.50
78.00	1	.50	100.00	.00

TABLE G.11. Bear Lodge--Central Block

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
31.00	1	.50	.50	99.50
33.00	1	.50	1.00	99.00
35.00	1	.50	1.50	98.50
37.00	1	.50	2.00	98.00
39.00	5	2.50	4.50	95.50
41.00	4	2.00	6.50	93.50
43.00	10	5.00	11.50	88.50
45.00	4	2.00	13.50	86.50
47.00	10	5.00	18.50	81.50
49.00	10	5.00	23.50	76.50
51.00	15	7.50	31.00	69.00
53.00	13	6.50	37.50	62.50
55.00	9	4.50	42.00	58.00
57.00	19	9.50	51.50	48.50
59.00	20	10.00	61.50	38.50
61.00	15	7.50	69.00	31.00
63.00	11	5.50	74.50	25.50
65.00	7	3.50	78.00	22.00
67.00	11	5.50	83.50	16.50
69.00	6	3.00	86.50	13.50
71.00	11	5.50	92.00	8.00
73.00	4	3.00	94.00	6.00
75.00	5	2.50	96.50	3.50
77.00	1	.50	97.00	3.00
79.00	1	.50	97.50	2.50
81.00	3	1.50	99.00	1.00
83.00	1	.50	99.50	.50
85.00	0	.00	99.50	.50
87.00	0	.00	99.50	.50
89.00	1	.50	100.00	.00

TABLE G.12.   Bear Lodge--South Block

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
32.00	1	.50	.50	99.50
34.00	1	.50	1.00	99.00
36.00	1	.50	1.50	98.50
38.00	1	.50	2.00	98.00
40.00	5	2.50	4.50	95.50
42.00	4	2.00	6.50	93.50
44.00	10	5.00	11.50	88.50
46.00	4	2.00	13.50	86.50
48.00	10	5.00	18.50	81.50
50.00	10	4.00	23.50	76.50
52.00	15	7.50	31.00	69.00
54.00	15	7.50	38.50	61.50
56.00	7	3.50	42.00	58.00
58.00	18	9.00	51.00	49.00
60.00	19	9.50	60.50	39.50
62.00	16	8.00	68.50	31.50
64.00	11	5.50	74.00	26.00
66.00	8	4.00	78.00	22.00
68.00	11	5.50	83.50	16.50
70.00	5	2.50	86.00	14.00
72.00	11	5.50	91.50	8.50
74.00	5	2.50	94.00	6.00
76.00	5	2.50	96.50	3.50
78.00	1	.50	97.00	3.00
80.00	1	.50	97.50	2.50
82.00	3	1.50	99.00	1.00
84.00	1	.50	99.50	.50
86.00	0	.00	99.50	.50
88.00	0	.00	99.50	.50
90.00	1	.50	100.00	.00

TABLE G.13. Conway Granite

Upper Limit	Observed Frequency	Percent of Total	Cumulative Percentage	Cumulative Remainder
38.00	1	.50	.50	99.50
40.00	1	.50	1.00	99.00
42.00	1	.50	1.50	98.50
44.00	3	1.50	3.00	97.00
46.00	5	2.50	5.50	94.50
48.00	8	4.00	9.50	90.50
50.00	14	7.00	16.50	83.50
52.00	15	7.50	24.00	76.00
54.00	17	8.50	32.50	67.50
56.00	20	10.00	42.50	57.50
58.00	22	11.00	53.50	46.50
60.00	14	7.00	60.50	38.50
62.00	12	6.00	66.50	33.50
64.00	15	7.50	74.00	26.00
66.00	7	3.50	77.50	22.50
68.00	9	4.50	82.00	18.00
70.00	8	4.00	86.00	14.00
72.00	7	3.50	89.50	10.50
74.00	2	1.00	90.50	9.50
76.00	5	2.50	93.00	7.00
78.00	5	2.50	95.50	4.50
80.00	2	1.00	96.50	3.50
82.00	2	1.00	97.50	2.50
84.00	2	1.00	98.50	1.50
86.00	1	.50	99.00	1.00
88.00	0	.00	99.00	1.00
90.00	1	.50	99.50	.50
92.00	0	.00	99.50	.50
94.00	1	.50	100.00	.00

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Federal Center
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J. A. Patterson
Director of Resource
  Assessment Operations
12th & Pennsylvania Ave., N.W.
Mail Stop 6521, RA 224
Washington, DC   20461

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Tucson, AZ   85721

W. M. Shaffer III
Acfg. Director
Fuel Cycle Project Office
Savannah River Operations
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